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J. W. RICHARDS, PH. D.President
E. F. ROEBER, PH. D.Editor

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Electrochemical Processes as Central Station Load Equalizers

At the Cornell meeting of the American Electrochemical Society, Mr. E. A. Sperry presented a paper on the possibility of using electrochemical processes as station load equalizers. The situation is briefly this. Central station engineers recognize the importance of filling out the valleys of their load curve, and to do this they could sell the current at very low rates, since any receipts would be "velvet." On the other hand, for the successful operation of any electrochemical and electrometallurgical processes, the chief requisite is cheap electric power. So far, the needs of the central station engineer and the electrochemist meet each other. But the trouble is that, at least in theory, the electrochemical engineer would have to take the electrical energy when the central station man cannot use it otherwise; and that he would have to be satisfied with how much the latter would see fit to give to him. Of course, no manager of an electrochemical works would undertake commercial work on such a basis; and the question is how much energy could central station engineers guarantee to deliver during a year. This is a question which can only be answered by experience on the basis of the daily load curves during a whole year. Some statistical data on what central station engineers are willing to do were given by Mr. Sperry's paper; and it is noteworthy to mention that, according to Mr. Sperry, when central station managers offer very low rates, they often demand the emergency privilege of cutting off the current. In the last end, of course, this, like any engineering problem, resolves itself into a question of dollars and cents. On the one hand, central station managers have to decide how low they can make the rate and still make a profit; according to Mr. Sperry, prices have been made which vary between 0.28 and 0.41 cent per kilowatt-hour (\$18 to \$26.50 per electric horse-power-year). On the other hand, electrochemical or electrometallurgical engineers must decide whether their processes can be operated intermittently with a profit. The discussion at Cornell seemed to indicate that this would be the case mainly with certain electric furnace processes, since electrolytic processes are mostly continuous.

* * *

The whole problem has again been discussed in an extremely able way by Mr. C. J. Russell, of the Philadelphia Electric Company, in a paper recently read before the Association of Edison Illuminating Companies. While it is an electric central station manager who has here the word, it is one of the very, very few central station managers of this country who has had practical experience with supply of electrical energy to electrochemical plants. It appears that the Philadelphia Electric Company has supplied energy for making bleaching liquor electrolytically, for making ozone by discharges through

air, and for making steel in the electric furnace. Concerning the last named application, a few notes of Mr. Russell may be given here in parenthesis, since electric steel is a subject dear to electrometallurgists and the plant in question is, although small, yet the first commercial induction-furnace plant for making tool steel in this country. As our readers know, the plant is at the Diston works and the Colby furnace is used. The furnace is of about 125-kw. capacity, and "several thousand pounds of each kind of high-grade steel of different composition used for cutting tools have been made in this furnace, and samples have been submitted to chemical and mechanical tests." * * * "The high temperatures attainable in the furnace render it possible to thoroughly remove all gases, and the steel is very fluid and still in the mould. The ingots are very dense and homogeneous, and tools made from them present a grain and silvery lustre unlike that of the ordinary crucible steel. The metal also has valuable characteristic qualities for the manufacture of high-grade tools requiring uniform temper. The results obtained have been so satisfactory that it is proposed to install a 5-ton furnace calling for about 750-kw. current capacity."

* * *

Mr. Russell brings out some new points which bear forcibly on the supply question. One is that the character of load diagrams of all central stations which have cultivated the electric power field has undergone a radical change in recent years. In such cases the load condition of morning and evening peaks which represented almost the total generated load has passed. In the present load diagrams the kilowatt area of such peaks as exist bear a much smaller ratio to the area of the daylight load. Now, in view of this fact, it is questionable whether a central station manager will undertake to make a contract for several years to deliver so and so much power at such and such hours to an electrochemical works. But would a contract for a shorter time be satisfactory to the latter? The second point, which, however, is related to the former, is the conviction expressed by Mr. Russell that for purposes of load equalization, it would only be possible to add such processes to any system as should be under the absolute control of the central station interests which were to supply them with electric current. It is an interesting fact that in a careful investigation, the results of which have just been published, Mr. Georg Dettmar, the general secretary of the German Association of Electrical Engineers, comes essentially to the same result. He does not speak of electrochemical processes operated from central station mains, but discusses, on the basis of extended statistical data, the different means of making smaller and medium-sized central stations more profitable. He shows how this can be done and has been done by undertaking outside work during hours of low load so as to keep the men fully occupied, but he shows that to do this successfully, the outside work, of whatever character it may be, must be in charge of the central station manager.

* * *

This final result may seem disappointing. It seems to rob the independent electrochemical engineer of a source of cheap power from which he had hoped much for the future. But,

disappointing though it may be, it is better to look squarely at a situation beforehand. Then there will be no disappointments later on. Naturally there is still a large field of possible developments, since when electrochemical processes become better understood by electrical engineers, they may be adopted by the central stations themselves. Local conditions alone can determine to what extent this would be profitable. Thus in certain districts it may pay a central station to make bleaching liquor or bleaching powder and caustic; in other districts this would be unprofitable, but other electrochemical products might pay. Finally, we might draw attention to a special point in Mr. Russell's paper, though it has only indirectly to do with our subject. Mr. Russell finds fault with the usual energy unit of electrochemical engineers, the "electric horsepower-year." He claims it is the unit of optimists, since estimates based on it are based on the assumption of the utilization of the whole power capacity during all hours of a year. Mr. Russell insists on the kilowatt-hour as being the only sensible unit. But this is another story, and we may come back to it another time.



The Mining Boom.

At present there is apparent in the North American continent unprecedented and extraordinary activity in mining. With the increased diffusion of scientific knowledge, due to the progress of technical schools, technical societies and technical journals, there has been a complete reversal of former procedure. Mining ores and extracting their values at a profit is becoming more and more an exact business where the certainty of profits can be determined and the risk eliminated as far as possible in what St. Paul called "this present evil world." In addition, there is the possibility of large profits from the development of unproven yet possibly mineral-bearing ground. All this appeals to the popular imagination, and the example of good profits made in mining companies by others has caused a widespread interest in mining companies. As John Hays Hammond said to an engineer acquaintance of ours: "Mining, if you know the game, or know a man who knows the game, is the best business in the world, but if you don't, it is the most dangerous." Staid old bank presidents and conservative business men are investing heavily. As all operations are done on a cash basis, if a crash should come it will not bring any great disaster, for the effect is gradually felt now on the New York Stock Exchange in the persistent selling of stocks and bonds by the public.

* * *

As Sir Isaac Newton said: "To every action there is an equal and opposite reaction," so too, stimulated activity in one branch of investment causes decreased activity in another. One very pertinent fact bearing on this subject is the influence of metallurgical progress on mining. The successful development of any mine depends on a successful reduction plant to handle its ores at a profit. Now the improved metallurgical processes have rendered mines once valueless of great value. This is true of all the domain of the "art of making money out of

ores." The cyanide reaction was the cause of a costly and bloody war in South Africa. The improvement in blast-furnace practice in handling soft Mesaba ores allowed the formation of the titanic "Steel Trust." Other metals, such as zinc and lead, would simply extend the list of illustrations. But without a doubt the most potent of all in widespread effects was the electrolytic refining of copper. This process has been the foundation, strong and solid, of modern electrical engineering, for without its "high-conductivity" copper the dynamo, telephone and transmission line were impossible as they exist now. Thus to America, Mr. Edward Balbach and his chief assistant, Mr. F. A. Thum, who originated electrolytic copper refining in this country in 1883, the mining and engineering world owe a debt of gratitude similar to that owed Bessemer.

The Recent Increase in Interest Rates.

In an editorial in our June number we pointed out several reasons for the increase in the rate of interest. We pointed out that one of the chief contributing causes was the increase in the productivity of capital, due to the irresistible progress of applied science. It was shown how applicable this was to the mining and smelting industries. The improvement in the blast-furnace practice has given actual value to the immense Mesaba ore deposit. This has resulted in the creation of securities of the United Steel Corporation. The open-hearth steel practice has likewise created wealth of phosphatic iron mines. This same course of events has extended throughout the metallurgy of the other principal industrial metals—lead, copper and zinc; and in general the effect can be traced in the entire commercial and industrial world.

The result we prophesied has come much quicker than was anticipated, and is seen in the fact that the staid old Bank of England has raised its discount rate to 6 per cent. At the present time in New York time loans on unquestioned security are being made at as high as 7 per cent. It is thought that this condition of affairs will continue for some time. Capital differentiated—to use the language of the calculus—is interest, the rate of growth of capital. And while returns to the borrower are so large, the capitalist demands and receives from the "entrepreneur" a large measure of payment. In the nature of things there must be some time a reaction. But the accurate diffusion of business and technical knowledge through the press, the celerity with which modern business is despatched, the improvement in farming methods, due to Secretary Wilson's propaganda, the solid advance in all engineering indicate that this day of reaction is far distant, provided only that the American people keep their heads. In other words, there is a great permanent movement due to a new, far-reaching and persistent cause, which has reversed the precedents of history and has lifted the world upward one step in her evolution.

The American Business Man.

In these days of unprecedented business activity in all circles, but especially that of mining and treating ores at a profit, which is the especial field of this journal, it may be well to analyze the characteristics of the guiding force of these commercial energetics. In America the "entrepreneur" and capi-

talist are often combined. Business judgment usually results in the possession of capital in the United States. With all due respects to the feelings and emotions of our socialistic friends, the management of any enterprise is the most important factor determining commercial success. The capitalist in this country is distinguished from his European brother in several marked features, partly due to his environment and partly due to his inherited qualities derived from his pioneer fathers.

* * *

One of the chief of these is the use of the imagination in holding mental conceptions clear in the mental horizon as a whole. What our German friends call "Geist," differentiates the American business man from all others. Hill, Morgan, Carnegie, Rogers, Harriman and Rockefeller all possess imaginative faculties, which if turned to literature or art, if these were the spirit of times, their inordinate ambition, and the environment suited, would raise these men to the first rank as artists. Cross and sordid as they may seem to the European, they are creative artists—business artists to be sure, but still creators into actuality of the figments of their minds. This most peculiar and seemingly and irreconcilable fact was brought out by Lefevre in his story, "Golden Flood." Naturally, their commercial imagination is always curbed by conservative judgment, but it is still the exercise of the imaginative faculty that is the reason for their commercial success. The next characteristic that is apparent in the American business man is the fighting spirit and persistency. Descendants of men and women who crossed the ocean to fight savages, not "Cooper Indians," to rear the civilization of democracy, these men possess the resolution of the pioneer. By selective emigration Europe is more or less barren of this spirit among her business men. Another trait of the American man of affairs is broadness of view and conception as can be expected in a country with Niagara Falls, the broad prairies and the Rocky Mountains. Ready adaptability to circumstances and environment is a fourth characteristic. This is seen best in the widespread use of the telephone, telegraph, typewriters, rapid transportation and other modern business machines. The use of these conveniences is the basic cause of enormous accumulation of personal fortunes in America.

* * *

We do not hesitate to criticize the American business man in two important particulars, lack of devotion to his family and lack of intelligent interest in the affairs of the nation. With his intense mental concentration—concentration is merely another name for imagination—the duties to his family and his State do not receive proper attention. The splendid gifts to science, education and art of many of our rich men attest that this fault is noted. But in this connection let it be said that for complete personal happiness there should be harmony between all duties. In the future, with free play of public opinion, this fault will be corrected somewhat. Lack of conservatism is a national fault, and the millionaire falls into the same category with the rest of us, for Americans are all consistent "bulls." Taken all in all, these men who have steered in the past generation our commercial ship are pretty good fellows, a bit peculiar in their persistency at the money game, but human like all of us, as Kipling said of "Tommy Atkins," "single men in barracks most remarkable like you."

Denver Meeting of the American Mining Congress.

(From Our Special Correspondent.)

On Tuesday, Oct. 16, the ninth annual session of the American Mining Congress was called to order at the Broadway Theater, Denver, Col., by the president, Judge J. H. Richards, of Boise, Ida. Among the members and delegates present were many representative men, mostly from the Western States, among them Governors Pardee, of California; McDonald, of Colorado; Brooks, of Wyoming; Cutler, of Utah; Ex-Governor Prince, of New Mexico, and a number of prominent Western mine owners and operators.

After the addresses of welcome by the Governor of Colorado and the Mayor of Denver, the morning session was devoted to brief addresses from delegates representing various States. The program of the afternoon session included the reading of several papers, the first of which, by Mr. E. W. Parker, of the United States Geological Survey, dealt with the loss of life in mining, and especially coal mining. Mr. Parker called attention to the fact that the number of lives lost in American coal mines is considerably greater than in Europe, and he pleaded for a more careful safe-guarding of the lives of the miners as well as for a rigid enforcement of the inspection laws.

Mr. Parker was followed by Dr. J. A. Holmes, of the United States Government coal testing plant at St. Louis, who spoke upon what European Governments are doing for mining. He described his observations in the principal mining districts of Europe, and referred particularly to the investigations undertaken by several European Governments in testing explosives and safety lamps; he announced the intention of the United States Government to begin similar experiments at its coal testing plant at St. Louis.

The next paper, by Mr. Thomas A. Rickard, the editor of the *Mining and Scientific Press*, was entitled "The Geological Distribution of Gold." The author showed that gold deposits are not confined to any particular formation, but that they are found in rocks of all geological ages.

In the following paper, "Mining and Mineral Resources of Utah," Mr. J. Dern, a well-known mine operator from Utah, drew an interesting picture of the resources of that State and the remarkable progress it has made during recent years.

The session on Tuesday evening was occupied by the annual address of the president and a paper by Mr. W. D. Brunton, of Denver, who introduced the draft of a proposed mine drainage district law which had been drawn up by a committee appointed for that purpose. The bill provides for unwatering all the mines of a mining district by a common drainage plant, each owner being proportionately taxed for the cost. The subject is an extremely important one for many districts.

The session on Wednesday, Oct. 17, was opened with a discussion of the mine drainage law, the final result being that the various speakers were invited to put their views in the form of amendments, to be presented to and considered by the committee responsible for the original draft, the committee being instructed to report to the Congress at the evening session.

Another very important subject was then introduced by Governor Pardee, of California, namely, the prevention of fraudulent mine promotion. Dr. Pardee drew attention to the injury to legitimate mining worked by these wildest schemes, and urged the adoption of the draft of a bill to be recommended for passage before the Legislatures of the various States of the Union, making the publishing of wilfully false or exaggerated statements a criminal offense. This paper was also followed by an earnest discussion, during which it developed that the convention was thoroughly in sympathy with the idea of suppressing mining frauds, but, that in the opinion of various speakers, some more fundamental legislation was needed, such as to establish State departments to pass upon the status of newly-promoted enterprises. Several resolutions looking toward that object were introduced.

The evening session was devoted to the discussion of the draft of the proposed mine drainage law, which was finally adopted with some minor modifications.

The session on Thursday, Oct. 18, was expected to be the most important on the program, as it was to be devoted to a discussion of the grievances of the ore producers against the so-called smelter trust—the American Smelting & Refining Co. The subject was introduced by State Senator E. De La Vergne, of Colorado, who drew attention to the intimate relation existing between mining and smelting, and complained about the absence of competition, the enforcement of arbitrary and unjust regulations, with especial reference to the Cripple Creek district, and the failure of the smelters in general to pay full value for the metals contained in the ore.

He was followed by Mr. F. Guiterman, the general manager in Colorado for the American Smelting & Refining Co., who endeavored to show that that company had competition in Colorado, that it had stimulated ore production in the various districts, and that its policy was justified from a business standpoint.

In the subsequent discussion various points were raised by the speakers, such as the question of the market quotations on which the settlement prices were based, the question of paying for small quantities of metals in the ore, etc.

The meeting then adjourned till the afternoon, when the discussion was to be continued. Owing to Mr. Guiterman's unavoidable absence, however, the discussion was closed and the afternoon devoted to the reading of several papers of general character, one of the most interesting being by Prof. Trapphagen, of the Colorado School of Mines, who dealt with the subject of smelter fumes and their effect on live stock and vegetation, with special reference to the suit pending against the Anaconda Co., of Butte. He showed that the effects were not as serious as they are usually regarded.

The session on Friday was devoted to the discussion and adoption of a number of resolutions and the election of officers, Judge Richards being re-elected president. Joplin, Mo., was chosen as the next meeting place. On Friday night about 150 of the members and delegates went to the Cripple Creek district as the guests of the Colorado Mine Owners' Association, where they spent Saturday in looking over the mines.

On the whole, the sessions were characterized by much earnestness of purpose. There was very little outing indulged in, and the social features were almost entirely absent. The participants realized that they were there to work and to discuss matters of vital interest to them and the whole mining world. The meeting can, therefore, be called a successful one, and has materially added to the standing and reputation of the American Mining Congress.

The Iron and Steel Market.

The great buying movement in iron and steel inaugurated about four months ago has had a wonderful course. If the orders and contracts prove good, and there is no reason to expect otherwise, the total of all finished steel sold is equal to full production until some time towards the end of the second quarter of next year. Of course, there is unsold tonnage for earlier delivery, balanced by sold tonnage for later delivery. The distribution is not entirely uniform among the different finished products, but it is more nearly uniform than would be expected, considering the varied uses to which steel products are put, and this encourages the idea that the reason for this heavy buying is psychological rather than physical. The era of "steady prices" and the scarcity of the fundamental raw material—pig iron—seem to have encouraged all consumers to buy very far ahead. There is physical ground for railroads buying steel cars far ahead, since the obvious needs of their traffic require the provision; there is no such ground in tin plate, yet the mills are booked through the first quarter

and well into the second quarter. Never before in the history of the tin plate trade has there been such buying for forward delivery in September and October. Two-thirds of the tin plate consumption depends upon the canning crops. The late canning season was not especially good, and nothing can be known of next season. Tin plate production is about the heaviest yet seen, and deliveries are fairly good.

Granting the psychological basis of much of the buying of finished steel products, the present acute scarcity of pig iron is readily explained. From time immemorial the alignment of the American iron and steel trade has been that there is an excess of steel-making capacity over pig iron capacity, and of finishing capacity over steel-making capacity. This, because there is less loss in operating a steel works to less than absolutely maximum capacity than with a blast furnace, and the same of finishing capacity as compared with steel-making capacity, while in addition it could not be expected that demand for the various finished steel products, from rails on through shapes, plates, merchant bars, wire and sheet products should be uniformly in proportion to the capacities installed. There are constant cross currents. Demand for rails may be relatively heavy and for sheet and wire products relatively light, and so on. Here has been a condition where demand has called for the operation of all finishing departments to their full capacity. Necessarily with these conditions a shortage in pig iron has resulted; had there been enough pig iron a secondary shortage would have been found in steel-making capacity.

Added to this there was a decline in pig iron production from March to the early part of September, from purely physical causes, but the ultimate buyers do not analyze what can be ascertained as to pig iron production as they should, and appear to have interpreted conditions as they worked out as indicating an increase in the ultimate consumptive demand rather than a decline in the production of the fundamental raw material. In March the United States was making pig iron at the rate of approximately 25,750,000 tons per year. Production declined steadily, until Sept. 1 it was at the rate of about 23,000,000 tons a year, through the humidity of the atmosphere and the necessity of relining an unusually large proportion of the furnaces. The rate of production is increasing rapidly, and barring unfavorable weather conditions leading, through railroad blockades and otherwise, to poor working, the rate could reach 26,000,000 tons before the end of this year, and 27,000,000 tons early next year. A number of new blast furnaces have been completed since last March, and others are to be completed shortly. Such results would undoubtedly change the whole aspect of affairs.

During the past month the best posted men in the trade have reached the conclusion, not always openly expressed, of course, that present conditions foreshadow the end of this remarkable movement, that next year will see declining prices and declining demand. The principal uncertainty is whether the full operations will last only to the middle of next year, or will be prolonged well through the second half.

PIG IRON.

Nearly all furnaces are behind on their contracts, and some consumers are buying prompt lots at fancy prices. Steel-making pig is fully sold through the first quarter, and very little is available for the second. Prices are nominal at about \$21, valley furnace, for Bessemer or basic. The shortage of foundry pig is especially acute in the Pittsburg district, and recourse is had to the South. Pittsburg consumers who ordinarily use only Northern iron have bought Southern iron freely. For instance, one consumer took over 1,500 tons of Alabama iron, for first quarter delivery, at \$17.00 and \$17.25, Birmingham, or \$21.60 and \$21.85, delivered Pittsburg, equal to \$20.75 and \$21.00 f. o. b. valley furnace. Most users of Southern iron are content, or even pleased, with the higher phosphorus contents of Southern foundry irons, but forced

consumers like the one mentioned are not, and are selecting the Southern irons, which are lower in phosphorus, so that a spread of 50 or 75 cents is sometimes found between Southern irons, which are usually on a parity. Virginia furnaces are quoting \$20.25, furnace, for No. 2 for delivery over first half, and will not sell for first quarter only. Valley foundry can be had at between \$19.00 and \$20.00, furnace, for second quarter, but earlier delivery cannot be quoted accurately.

BILLETS AND SHEET BARS.

The large steel mills, being busy in their own finishing departments, and having some crude steel contracts on which they are behind, are not sellers. They are continually turning down large inquiries. A very large tonnage of billets could be sold at present nominal prices of \$28 to \$29 for Bessemer and \$29 to \$30 for open-hearth, but the inquirers could not afford to pay a much higher price, as prices on finished materials have been held down. Steel mills are very hard pressed to make outputs, and there is more laxity in analyses. It has long been the case that Bessemer pig, outside the phosphorus limit, has been blown, resulting in steel from .10 to .12 in phosphorus, for uses where the defection would not be noticed, and of late there is ground for believing that considerable tonnages of relatively low phosphorus basic iron have been used in the converter. Throughout the industry the rules against phosphorus are being relaxed. Thus, in "low phosphorus pig," used for special acid open-hearth work, the commercial definition has been raised from .035 to .040. Some iron as low as .025 can still be made, but more iron had to be let into the class. The Republic Iron & Steel Co. has made some important sheet bar contracts for first half delivery, chiefly with tin mills, on the basis of about \$30.00 f. o. b. Pittsburg. The company naturally finds it much more profitable to turn its surplus steel into sheet bars at such prices than into billets or rails at \$28, in its convertible mill, and it is practically out of the rail market.

FINISHED MATERIAL.

The National Tube Co., a subsidiary of the United States Steel Corporation, on Oct. 13 advanced merchant pipe two points, or about \$4.00 a net ton, and casing about \$2.00 a net ton, establishing the market on the basis of 79 and 5 off list on merchant pipe 3/4 to 6 inches inclusive, in large lots. While the steel corporation has been uniformly opposed to any price advances at this time, in order to keep the brakes on the market, the independent pipe mills claim this advance was forced by their refusal to book ahead at the low prices, throwing such a tonnage of business to the leading interest that it had to yield.

Other finished material prices are unchanged, except that a concession of 5 cents a hundred on wire products has been withdrawn, the official price remaining unaltered. Prices are, therefore, as last quoted:

Beams and channels, 3 to 15 inches, inclusive, \$1.70; over 15 inches, \$1.80; tees, \$1.75.

Plates, tank quality, 1/4 inch and heavier, 100 inches wide and less, \$1.60.

Steel bars, \$1.50, base; iron bars, \$1.60, base.

Sheets, 28 gauge, box annealed, one pass cold rolled, \$2.50; galvanized, \$3.55; blue annealed, No. 10 gauge, \$1.75.

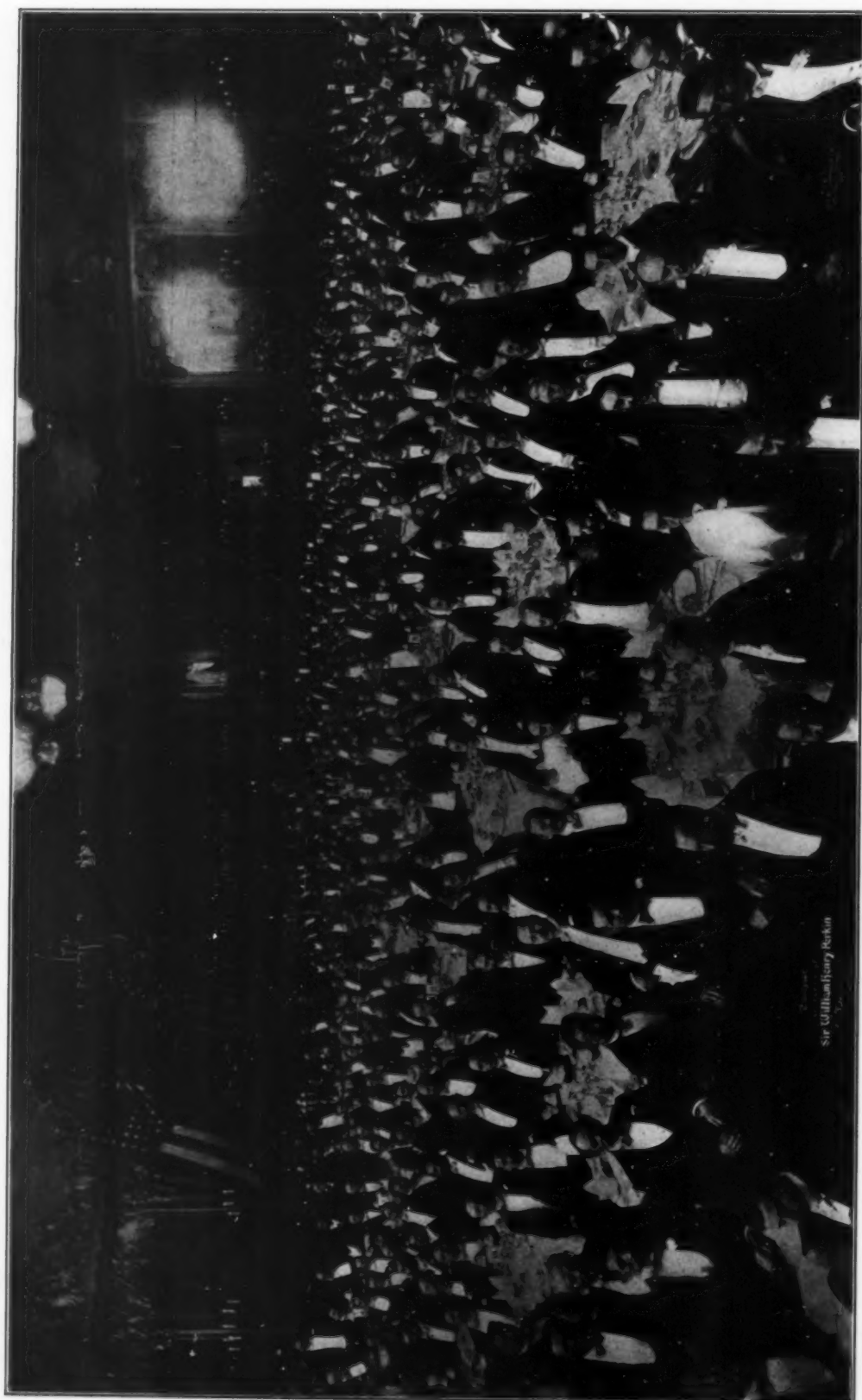
Plain wire, \$1.70; wire nails, \$1.85, both base.

Tin plates, 100-pound cokes, \$3.75 per box.

Above prices are for carload and larger lots, f. o. b. Pittsburg; on sheets and tin plates a concession of 5 cents is given; wire concessions are claimed to be entirely withdrawn.

COKE.

There has been a spectacular buying movement in furnace coke for next year. Early in the month the regular buyers of Connellsville furnace coke were hesitating about paying \$2.75 to \$2.85 on their contracts for next year. The H. C.



PHOTOGRAPH TAKEN AT DINNER TENDERED TO SIR WILLIAM PERKIN.

Sir William Henry Perkin

Frick Coke Co., which handles all the steel corporation coke, and has heretofore been able to meet all requirements except in extraordinary conditions, suddenly appeared as a buyer and made large contracts on the advancing market, from \$2.85 on up to \$3.10 for standard Connellsville furnace coke, taking altogether between 400,000 and 500,000 tons of Connellsville, about two-thirds for first half and the balance for second half, besides buying liberally of West Virginia coke. Other consumers are turning more and more to coke outside the old Connellsville region. The coking coal resources of that region are well known; at the present rate of exhaustion they will last about twenty-five years. The rapid growth in production appears to be checked, at any rate the number of ovens has been stationary for a year or more. An acute scarcity of coke cars is on all hands expected this winter.

The Coal-Tar Industry Jubilee.

The fiftieth anniversary of the invention of the dyestuff mauve by Sir William Henry Perkin has been duly celebrated as an event of international importance. England had its celebration during July, and many scientists and chemical manufacturers from other countries gathered at that time in London to pay their respects to Sir William.

NEW YORK CELEBRATION.

America followed in October with its own celebration, which culminated on Oct. 6 in a banquet in New York City, which was tendered to Sir William at Delmonico's, and was attended by some 400 American chemical manufacturers and scientists.

Dr. C. F. CHANDLER was a genial toastmaster. In opening the after-dinner speeches he proposed a toast to the President of the United States, the King of England and the Emperor of Germany. It has been said that in the coal-tar industry these three countries form a triple alliance, Great Britain being the seat of the starting of the industry, Germany the chief manufacturer of dyestuffs, and America the country of coal-tar distillation and a large user of colors.

In his speech, Dr. Chandler referred to the scheme of establishing a chemical library in New York, to be known as the Perkin Library. This is intended to cover the complete field of theoretical and applied chemistry, with duplicate sets of all books, one for consultation at the library, the other for circulation among American chemists. The scheme is quite pretentious, since the library will be under the supervision of two trained chemists, and is intended to be maintained not simply as a library, but also somewhat as an information bureau for American chemists on all questions of a chemical character.

The president of the Board of Aldermen of New York City, Mr. P. F. McGOWAN, as the representative of the Mayor, Mr. G. P. McCLELLAN, welcomed Sir William in a humorous speech to the city of New York.

Dr. HUGO SCHWEITZER then presented what was undoubtedly the most elaborate speech of the evening. In a 15-minute address he cleverly condensed in concise form the vast material, which, as he said himself, could have been covered more easily in an address of 15 hours. After speaking of the way in which Sir William, as an 18-year-old boy, discovered mauve, he emphasized the enormous difficulties which he had to overcome later on in manufacturing the dyestuff on a commercial scale and in inducing dyers and printers to adopt it. The latter were extremely conservative, and would use nothing but natural colors with the formulas inherited from their grandfathers. It has been almost forgotten that Perkin was the first to introduce the method of dyeing silk in a soap bath, which is commonly employed to-day for all artificial dyestuffs, and that he and Pullar first made use of the mordanting of cotton with the insoluble, inorganic compounds of tannin.

After these initial commercial difficulties had been overcome

by Sir William, it was comparatively easy to introduce other coal-tar colors. But Perkin also paved the way for the discovery for the later coal-tar colors by creating commercially aniline and benzol, which up to his time had only been laboratory curiosities. Of the three methods available for obtaining aniline he selected as the most promising the reduction of nitrobenzol, made by nitration of coal-tar benzol, and the production of aniline from this source led shortly after the discovery of mauve, to the discovery of magenta, which opened up a new and immense field for this industry.

To-day about 2,000 individual dyestuffs are known, giving the whole range of the colors of the rainbow, and complying with every demand of taste, fashion and stability. They surpass in beauty and brilliancy the colors supplied by nature, and contrary to the impression prevailing among the public, the shades obtained with some of them are faster to the influence of time, light and chemicals than the fastest which nature produces. The greatest triumph of this branch of the industry was the artificial production of alizarine and indigo. In the technical production of alizarine, Sir William has played a prominent part. Coal-tar colors, however, are not only used for the dyeing of textile fibers, like wool, silk, cotton, linen, jute, ramie, etc., but for a host of other materials.

"The lady's hair is gray, or of a hue not fashionable at the time—coal-tar colors will assist her in appearing youthful and gay. In eating the luscious frankfurter your soul rejoices to see the sanguineous liquid oozing from the meat—alas, coal-tar colors have done it, and friend Wiley can prove it. The housewife selects a bright green broom, on account of its anticipated good wearing quality, but finds to her sorrow that coal-tar colors furnished the freshness. The product of the hen is replaced by yellow coal-tar colors in custard powders, and butter is colored yellow when the dyestuff laboratory of the cow is on a strike. Leather, paper, bones, ivory, feathers, straw, grasses are all colored, and one of the most interesting applications is the dyeing of whole pieces of even the bulkiest furniture by dipping them in large tanks containing the dyestuffs, which transforms the wood into walnut, mahogany at your command, as carried out in our big factories in Grand Rapids and elsewhere."

In the further course of his address Dr. Schweitzer referred to the medicinal properties of the coal-tar colors, and gave an amusing account of the way in which the beneficial effects of some coal-tar remedies were discovered. He then spoke of saccharine as a substitute for sugar, and then dealt with the industry of artificial perfumes. He finally referred to the prominent part which coal-tar colors and preparations now play in the reproductive arts, such as printing, writing, photography, etc. Curiously, that problem which was the incentive of Perkin's research, and which started the whole coal-tar industry—the artificial production of quinine—remains unsolved up to the present day.

In conclusion the speaker referred to the enormous influence which the development of the coal-tar industry has had on other chemical industries. "The coal-tar industry gave us our modern chemical institutes, the wonderful equipments of which were first utilized in the laboratories of the factories, and, above all, it gave us the intimate coöperation of technics and science, which is in fact at the root of all this magnificent success."

Dr. W. H. NICHOLS, president of the General Chemical Co., then presented to Sir William Perkin the first impression of the gold medal to be known as the Perkin medal, and to be given hereafter annually to the American chemist who has most distinguished himself by his service to applied chemistry.

Mr. ADOLF KUTTROFF, who has been the pioneer of the coal-tar industry of the United States, presented to Sir William a silver tea service of eight pieces, as a token of gratitude from his American friends. (Sir William later on said that the present was not only beautiful but also a very useful and a very suitable gift for a total abstainer.)

which ceased in 1873. After this date Sir William occupied himself purely with scientific researches.

The banquet was undoubtedly one of the most successful and most representative ones ever held by scientists and engineers in this country, and not enough credit can be given to the committee of fifteen who made all the arrangements. Dr. C. F. Chandler was the chairman, Mr. A. Kuttroff the treasurer, and Dr. Schweitzer the indefatigable and ever resourceful secretary.

On the evening of Oct. 9 a reception was given to Sir William at the Chemists' Club.

BOSTON CELEBRATION.

The chemists of the New England States tendered a dinner to Sir William on Oct. 11, at the Algonquin Club, in Boston. The attendance was about 170. The celebration was under the direction of Mr. F. E. Atteaux, while Prof. H. P. Talbot acted as toastmaster.

Among the speakers were Governor Curtis Guild, Jr., United States Senator Henry Cabot Lodge, Dr. Henry S. Pritchett and Mr. W. Whitman. Dr. W. H. Walker, of the Massachusetts Institute, presented to Sir William a silver punch bowl. The reply of Sir William was in general the same as at the New York dinner.

The British Columbia Zinc Report.

The Department of the Interior of Canada exercises a paternalistic care of the mining and smelting industries. Not only have iron and steel companies been liberally subsidized by the Dominion at their outset, but realizing the throttling of the iron industry by lack of adequate coke supplies, they made a most careful investigation of the possibilities of electric iron smelting and steel refining. This was published in the well-known comprehensive report of the commission with Dr. EUGENE HAANEL, Superintendent of Mines, as head, which has had such a stimulating effect on industrial developments.

It has been long known that there were extensive bodies of zinc-silver-lead sulphides in the "Slocan District" and elsewhere in British Columbia. In past ages ore has been shipped to Belgium and to the Kansas zinc plant at a freight rate of about \$10 per short ton.

To secure the proper technical knowledge bearing on the zinc industry, the department engaged a commission of engineers from the United States. The chief of these was Mr. W. R. INGALLS, a distinguished author on the metallurgy of zinc, well known because of his two comprehensive treatises on the subject. Under Mr. Ingalls were Mr. Philip Argall, of Denver, and Mr. A. C. Gardé, of Denver, both of whom have the reputation of being careful engineers. Mr. Henry E. Wood, of Denver, conducted the tests at his ore-testing plant.

The result of their extended and exhaustive investigations is a book of nearly 400 pages. The subject has been studied from the commercial, metallurgical and mining standpoints. In the question of the valuation of ore deposits and ore prices, the commission has recourse to the formula used by the Belgian zinc plants, which this journal recommended in an editorial in our March number. This method is being somewhat followed in the United States. Spelter and Joplin statistics are given in full, so that the volume forms a ready supplement to previous volumes of Mr. Ingalls.

Especially to be recommended is the section devoted to ore dressing. The magnetic, electrostatic, "flotation," wet processes are elucidated. Numerous tests were made on the ores of the several camps at Denver and the grade raised in cases to over 50 per cent zinc. The various separators are described and even the patent situation investigated. In fact, there is no work extant in the English language which better describes the up-to-date methods of concentrating complex zinc-lead sulphides than this.

Mr. Ingalls gives considerable space to the consideration of the question of handling zinc ores or zinc concentrates in the electric furnace. His conclusions are that there is absolutely no chance for electric smelting unless power is generated at a figure of \$12 per horse-power-year or less in an hydro-electric plant. He also points out vaguely certain metallurgical difficulties in the way of the practical accomplishment. We discussed fully the subject of electric zinc smelting in an editorial in our June issue of 1905.

Our present views are somewhat at variance with those of Mr. Ingalls. While we believe that it is a very difficult problem, yet we consider that intelligently directed hard work will solve the problem in the next ten years, and that the electric furnace will sooner or later modify the metallurgy of zinc and possibly revolutionize it.

Mr. Ingalls is not an electrical engineer, for, had he been so, he never would have overlooked on page 335 the use of the phrase "with a current of 120 volts." Electric current is measured in amperes at a pressure of so many volts, and accordingly we can assume that he is not as familiar with electric furnace design and practice as are those who are electrical engineers.

Concerning the efficiency of the electric furnace, Mr. Ingalls declares that "50 per cent is probably as high as should be reckoned at the present time." This is about right for the so-called induction furnace. With all due respect to the excellence of this special type of furnace for certain work, we do not think that any electrochemical engineer would recommend it for zinc distillation. It is a fact that other types of electrical furnaces show efficiencies of 80 to 90 per cent. In view of this fact, we may say that there is as great danger in making ultra-conservative positive statements as in making random guesses. We believe that time will prove the truth of our assertions.

Taken all in all, we can praise the careful and painstaking work of the commission as well as the discerning paternalism of the Canadian Government in providing proper funds for so complete and accurate study of the best way to develop the mineral resources of the country. The United States Government would do well to emulate the example of Canada. Perhaps the proposed Bureau of Mines at Washington will effect this happy result.

We consider that the British Columbia zinc-silver-lead camps have a promising future, and with adequate concentration and smelting facilities will be large producers of zinc and lead for years to come. To this end the work of the commission has contributed much.

CORRESPONDENCE.

American Electrochemical Society.

To the Editor of *Electrochemical and Metallurgical Industry*:

SIR:—Please permit me, as a New York member of the American Electrochemical Society, to point out a few facts which should be generally known to understand the undeniable lack of success of the recent New York meeting of the Society.

The members which constituted the local New York committee did not hear of their appointments until three weeks before the meeting. In the short time which remained they did all they could, and the program they arranged was undoubtedly attractive. But the arrangements were, naturally, finished so late that no proper announcement could be made in advance. Otherwise the excursion to the splendid Maurer works should have attracted a great many more members.

No printed "definite" program of the papers to be presented at the meeting was available until the Saturday immediately preceding the Monday on which the meeting began.

The best arranged meeting will fail if not properly announced and advertised. Engineering societies should learn to properly value publicity.

A NEW YORK MEMBER.

NEW YORK MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY

The Autumn meeting of the American Electrochemical Society (its tenth general meeting) was held in New York City on Monday and Tuesday, Oct. 8 and 9. These dates had been selected so as to enable visiting members to attend the dinner given to Sir William Perkin on Oct. 6 and the reception on Oct. 9. We report at another place of this issue on these functions.

The meetings of the American Electrochemical Society were held in a lecture hall in Havemeyer Building, Columbia University.

MONDAY SESSION.

The first session was called to order by President Carl Hering, on Monday, Oct. 8, at 10 A. M.

Dr. C. F. CHANDLER, the distinguished professor of chemistry at Columbia, made a most cordial speech of welcome. While the American Electrochemical Society is the youngest chemical society, he thought it would prove one of the most important. Electrochemical methods have already worked revolutions in many chemical processes. It is a really great pleasure now to visit one of the great electrochemical works and to compare the new method of working with the old chemical one concerning simplicity, ease and convenience for the workman. For instance, to manufacture caustic soda, how easy and simple is the electrolytic method compared with the old reverberatory furnace process. Dr. Chandler then pointed out the complete revolution brought about by electrochemistry in the manufacture of aluminium. He referred to some details in the electrolytic process, which indicate in an interesting manner the simplicity of the operation and the better protection of the workingman. The molten bath is, of course, very hot. Mr. Hall found that to protect the workingman against the heat it was sufficient to put a layer of pulverized coke on top of the bath. The alumina is then thrown on it and dried by the heat. During operation the bath, which is a mixture of alumina in cryolite, becomes impoverished in alumina, so that at intervals more alumina has to be added to the bath. To find out when this becomes necessary the simple expedient is used of putting a 20-volt lamp between anode and cathode (in shunt to the bath). As long as the bath contains sufficient alumina no current passes through the lamp which remains black, since only about 7 volts are required for electrolysis. When, however, the alumina is being consumed, the voltage rises (since the electrolysis of cryolite would require 25 volts) and the lamp lights up. This is a sign to the workman to add new alumina. In almost all electrochemical plants we see the wonderful effects of simplifying metallurgical operations by electrochemical methods, and of saving the wear and tear on human beings who work the process.

Dr. Chandler finally referred to the fact that whenever a new society is founded the older ones feel somewhat sore about it. It was so with the Perkin jubilee. When the English committee which arranged the big celebration in England, heard of the intention of American chemists to hold their own celebration in New York, they felt sore. Later on New York felt sore when it was reported that Boston would also have its own celebration. Finally everything turned out to the greatest mutual satisfaction, and there was room and glory enough for all. It is the same with the formation of new societies—the more societies the better for the cause of science.

Mr. Carl Hering, as president, expressed the thanks of the Society to Dr. Chandler for his speech of welcome, and to Columbia University for the courteous hospitality extended to the Society.

CONCENTRATION CELLS.

The first paper, by Dr. HENRY S. CARHART, discussed the formula for the Helmholtz concentration cell. In the absence of the author the paper was read by Dr. Bancroft. The author

referred to his former statement that concentration cells are devices for converting the heat of their surroundings into electrical energy. The well-known Helmholtz equation for the e. m. f. for any reversible voltaic cell becomes of specially simple form if the energy of the chemical reaction is zero. Dr. Carhart maintains that in simple concentration cells—in which two electrodes of the same metal are immersed in two different concentration of a solution of a salt of the metal—the condition is approximately fulfilled that the heat of reaction and of the dilution is negligible. In this case the Helmholtz equation assumes the form that the e. m. f. equals the absolute temperature, multiplied by the temperature coefficient of the e. m. f.

Dr. Carhart showed that the same equation may be derived from Nernst's formula for the e. m. f. of a concentration cell as function of the two concentrations. The results were confirmed by experiments with two concentration cells, one containing silver electrodes in silver nitrate.

The paper was briefly discussed by Messrs. Hering and Bancroft.

VISIBLE MIGRATION OF PARTICLES.

President CARL HERING presented a paper with this title, in which he first called attention to some peculiar facts observed in electrolysis. In the electrolytic deposits of iron the deposited iron sometimes contains carbon from the cast iron or steel anode. In the electrolytic deposits of alloys the deposit on the cathode sometimes contains some oxide. The deposit of silver in the silver voltmeter sometimes exceeds that required by Faraday's law. When a bath is turbid with suspended particles, the deposits on the cathode sometimes become rough, due to the foreign materials becoming enmeshed by the molecules of the metal which is being deposited.

These facts suggested to the author the possibility of a molar migration of suspended particles under the action of the electric current, essentially different from ionic electrolytic migration.

During the past winter Mr. Hering and Dr. E. F. Northrup made some microscopic researches in connection with the very interesting, curious and still unexplained Brownian movements, those persistent and untiring movements of minute, lifeless bodies which are perhaps still the closest approach to an apparent perpetual motion yet observed. These small but molar particles (which for convenience of reference were nicknamed "brownies") were subjected to the influence of an electric current in the surrounding liquid, or to charged electrodes in insulating liquids, and it was then noticed that they wandered across the field from one electrode to the other.

Their motion was very decided; it was along the lines joining the electrodes, that is, from one electrode to the other, showing that they were attracted by one or the other electrode. Their direction of motion depended upon circumstances; most of them traveled with the current, while some traveled in the opposite direction, namely, toward the anode; but each particle had its characteristic direction of motion, relatively to the current, which never changed, at least so far as the experiments showed.

In some mixtures of particles of two different materials, those of one kind traveled with the current and those of the other against it; when the current was stopped or reversed their direction of motion was stopped or reversed with great promptness, like an aperiodic galvanometer; they acted as though they had no inertia; that is, as though the moving forces were immensely greater than those due to their inertia. They obeyed the current as promptly as a body of well trained soldiers obey their officer.

Their speed increased with the current or the difference of potential between the electrodes, and seemed to be approxi-

mately constant for constant electrical conditions; it was under certain conditions so great that the particles raced across the field in a few seconds (actual speed 1 foot in 4 hours); the smaller particles, as a rule, traveled faster than the larger ones. In some cases an extremely small difference of potential sufficed to move them.

These motions were so positively related to the electrical conditions that it suggested the idea to Dr. Northrup of making it the basis of a sensitive electrical measuring instrument for special purposes.

A peculiarity of these particles while thus in motion, was that they repelled each other during their migration; they rarely if ever quite touched each other; when two particles traveling in opposite directions approached each other, they deviated their paths as they passed each other, like two wagons meeting on a narrow road. They acted as though each were surrounded by an invisible sphere of some repellant force like a very elastic jelly. When electrodes were used in non-electrolytes the particles traveled up to their respective electrodes, and crowded together around them like flocs of ice in a running river, but apparently not quite touching them.

The motion takes place in electrolytes as well as in insulating liquids; it is not limited to solid particles but takes place also with liquid particles suspended in a non-mixible liquid, like the little spheres of liquid fat in milk; milk, under certain conditions, even shows the rather rare phenomenon of two kinds of particles traveling in opposite directions in the same liquid. Almost all the particles which were examined traveled in the same direction as the current, namely, from the anode to cathode.

The author suggested that since some colloids have been known to travel bodily when a current is passed through them, and since many of the colloids show Brownian movements, the phenomenon described in his paper might be of the same nature.

In the extended discussion of the paper which followed, Prof. Bancroft questioned whether the different facts mentioned in the introduction of Mr. Hering's paper would be correctly explained as a result of colloidal movement under the action of electric current. In respect to the interesting observations of Mr. Hering, he thought that what had been watched under the microscope was in reality the opposite to electric endosmosis. An electric current always causes a relative motion of a diaphragm and a liquid. Whichever is movable moves. When we have a solid stationary diaphragm in a liquid the liquid moves, and we have the phenomena of electric endosmosis as first studied by Quincke. On the other hand suspended particles in the liquid represent a movable diaphragm, and will then move under the action of the current in a direction opposite to that of the movement of the liquid.

Mr. C. E. Acker briefly called attention to a recent German patent of J. A. Nussbaum (see our October issue, page 416 and page 379).

Mr. Hering, in reply to Prof. Bancroft, did not see that any other satisfactory explanation of the occurrence of carbon in electric iron deposits had been given than that given on the basis of his paper. With respect to the patent of Nussbaum, which he called quite ingenious, he explained that its quintessence is to add to the electrolyte a very small quantity of such a colloid which will travel with the current towards the cathode. Under these conditions (Hering's explanation is somewhat difficult from and somewhat more plausible than that of Nussbaum) the colloidal particles will travel with the current and at such places where there is a tendency of forming a tree, resulting in a high-current density, the colloidal particles will accumulate and will form a protective diaphragm. The lines of current will thereby be shifted to other places at which the current density was less before, and so on. In this way the floating colloidal diaphragm will tend to produce a uniform metallic deposit.

Mr. E. A. Sperry called attention to the fact that Mr. Betts

has successfully used the same method for years in his lead-refining process on a large scale, and asked what was the difference between Betts and Nussbaum. Prof. Bancroft answered that Betts uses glue or gelatine, because he has found it effective for getting a smooth deposit of lead. Nussbaum would use glue because it travels toward the cathode with the current, and would therefore be effective. Nussbaum's patent is an outcome of investigations why the addition of some colloids is beneficial while others have no effect. It was found that those which are beneficial travel with the current, the others in the other direction.

LABORATORY APPLIANCES FOR ELECTRIC FUSION AND OTHER WORK.

Mr. S. S. Sadtler then presented a paper on some very simple laboratory devices for doing electric fusion work. The author has used apparatus ordinarily found in a chemical laboratory for doing quite a range of electrochemical work, and in a number of cases, when he had exhausted his laboratory facilities, larger furnaces, etc., were installed on terra firma.

The current used for the different experiments referred to was from the city lighting service, the voltage being 220 between the outers of the three-wire system. The voltage available between an outer and the neutral was, therefore, 110. Both 220 or 110 volts could thus be obtained with very little trouble. A very simple bank of lamps was used to vary the current. Six stationary sockets were fixed to a board and wired in multiple, with binding posts at one end for connection to the work. The current could thus be varied by putting in more lamps, and varying the capacity of the same from 1/5 amp. to 5 amps., the 1/5 amp. being approximately obtained with the use of one 220-volt, 16-candle-power lamp, while the 5 amps were obtained by the use of six 110-volt, 32-candle-power lamps, which gave nearer to 5 1/2 amps. In this way all the current desired was quickly obtainable and without danger of short circuiting.

For carrying out a quick experiment to see whether a metal is deposited, or to get a rough idea of the state of fluidity of a proposed bath, a Chaddock gas burner is found serviceable, upon the top of which a small Hessian crucible or one of iron or nickel can be rested, in which the molten electrolyte can be contained, and the anode, a rod of graphite, or whatever is desired, can be dipped into it from a clamp which makes electrical contact, and for a cathode, a very simple and effective device is made as follows: A wire is bent into a spiral at one end, which fits in the bottom of the crucible and more or less covered by the separated metal, and the surface of the wire contained in this spiral is made proportional to the desired current density. The stem of the spiral comes up the inside of the crucible, and is cemented to it by a plastic mass, which, in some cases, could be fire-clay or carborundum powder and silicate of soda, which works satisfactorily for many fusions of a neutral or acid character.

When a larger experiment is to be performed a Buffalo dental furnace, with a large gas blast lamp or gasoline lamp for purposes of heating, was found very satisfactory by the author. In the top of this the crucible can be inserted, of proper size, with the same spiral arrangement for cathode. In some cases, where an iron cathode is wanted of considerable surface, a large iron crucible or iron body of an iron still of about a quart capacity may be used in the furnace.

In some experiments carried out by the author, in which certain gases were passed over a solid at definite temperatures, a tube furnace was employed, which was very effective for the purpose.

In looking up the proper size of platinum wire for electrical heating, it was almost impossible to decide upon the size and amount of wire that would be required, and it was not until a trial was made that a fair idea could be gotten. The author gave the following specifications for this furnace:

The tube consisted of Royal Berlin porcelain, of about

1/2-inch inside diameter, and 2 feet long. Around this, allowing about 8 inches free at each end, was wrapped platinum wire of No. 28 gage. It took 10 feet of wire to cover the body of the tube, allowing about 1/8 to 3/16-inch space between the folds of the wire. As the platinum wire was wrapped tight and clamped at the ends with heavy copper wire, the free ends of which were used as electrical terminals, it was not found necessary to put in any insulating material to keep the wire apart. Asbestos cord could have been used for this purpose.

A small section of 3-inch galvanized piping, such as that used for rain spouts, was used for the outside shell, and between this and the tube a mixture of magnesia and asbestos was packed, sheet asbestos being put on the ends and fastened thereto with silicate of soda wired.

A variety of rheostats may be used for varying the current. The author used one of small discs of carbon, the resistance of which was varied by adjusting the pressure by means of a screw. Temperatures varying from 600° or 700° F. to 2,000° were used by the author, and undoubtedly considerably more could have been obtained. The pyrometer used, however, only registered to 2,000.

The control of temperature was gotten by the use of an ammeter in circuit, and a pyrometer, the fire end of which was encased in a small porcelain tube, which ran in the larger one and held at one end firmly in place by means of a rubber stopper. The use of an ammeter is quite essential to the successful control of an experiment, as the pyrometer, especially one encased in a porcelain tube, does not register a change of temperature quickly enough, and too high or too low temperatures would be the result where more time would be desirable.

COPPER CATHODES IN NITRIC ACID.

A paper by J. W. TURRENTINE on this subject was read in the absence of the author by Prof. Bancroft.

Attention is called to the fact that nitric acid in dilute solutions undergoes reduction quantitatively at the cathode. This method has been recommended for the quantitative determination of nitrates. This reduction is equivalent, of course, to an action by nascent hydrogen, liberated at the cathode, on the nitric acid, according to the equation:



Nitric acid in acting on copper, on the other hand, is reduced not to NH_3 , but to nitric oxide, as indicated by a free evolution of brown fumes. In other words, copper reduces HNO_3 to NO and not to NH_3 . It reduces nitro-benzene to aniline. Why does it not reduce HNO_3 completely? Why does the reduction stop at nitrites?

These facts appear to contradict Bancroft's general law of the parallelism between electrochemical and purely chemical action and a series of experiments were made to explain this discrepancy. The first clue was found when experiments showed that HNO_3 could not be reduced quantitatively to NH_3 in the presence of Cu ions, and that the reduction stops at the nitrite stage.

The chief results of the author's experiments are that, when copper dissolves in nitric acid the anode reaction is the formation of copper nitrate and not of nitrite. When copper dissolves in nitric acid the cathodic reaction is the reduction of the nitric acid. The chief reaction product is NO when there is copper as ion in the solution, and NH_3 when there is practically no copper as ion in the solution. When a solution of copper nitrate and nitric acid is electrolyzed, with a copper cathode, nitric oxide is formed at the cathode. When a solution of nitric acid is electrolyzed, with a copper cathode, ammonia is formed at the cathode. The apparent discrepancy between the chemical and electrochemical action of copper on nitric acid is, therefore, due entirely to a difference in the conditions under which the two sets of reactions have been performed. The paper was discussed by Dr. Thatcher and Prof. Bancroft.

ALUMINIUM MAGNESIUM CELL.

A paper by Messrs. G. H. COLE and H. T. BARNES, on an aluminium-magnesium cell, was read in the absence of the authors in abstract by Prof. Barnes.

The authors describe a series of experiments with a cell in which a magnesium and an aluminium electrode were used with potassium alum as electrolyte. This acted as a galvanic cell with the current passing in the cell from aluminium to magnesium. The most interesting result is that when hydrogen peroxide was added as a depolarizer, or oxygen bubbled through the cell, an e. m. f. of 2 volts could be fairly well maintained.

Since aluminium and magnesium are so close together, the high e. m. f. is remarkable, but no explanation of it was offered by the authors.

In the discussion which followed Prof. Loeb suggested that the magnesium might decompose the water, and the cell might be after all a gas cell.

MELTING CURRENTS OF CRYOLITE-ALUMINA MIXTURE.

A paper by Mr. FRANCIS R. PYNE was then read in the absence of the author by the secretary, Mr. S. S. Sadtler. The melting points of different mixtures of cryolite and alumina, containing between 0 and 20 per cent of alumina, were determined by plotting the cooling curves with the aid of thermocouples. The chief results are given in the following table:

Cryolite %....	100	97	96	95	94	93	92	90	85	80
Alumina %...	0	3	4	5	6	7	8	10	15	20
Melting point..	1000	974	960	915	960	982	992	980	994	1015° C.

The melting point is a minimum for a mixture containing 5 per cent of alumina. The drop just behind 8 per cent seems to indicate the formation of a compound.

DOUBLE DECOMPOSITION OF ZINC SULPHATE AND SODIUM CHLORIDE.

In a paper by Mr. P. B. SADTLER and Dr. W. H. WALKER, an account is given of an investigation, the object of which was to find a cheap and easy method of making zinc chloride from sulphate. This is accomplished by using saturated solutions of common salt and zinc sulphate and chilling them to a certain point, which results in the crystallization of glauher salt, leaving zinc chloride. Under proper conditions an efficiency of almost 100 per cent may be obtained. In the explanation of the theory of the process, the phase rule was found to be of much usefulness, but this part of the paper was only briefly read in abstract.

DISTRIBUTION LAW.

A paper by Dr. H. E. PATTEN, of the Bureau of Soils, Washington, entitled "Some Factors Affecting the Distribution Law," was read in brief abstract by Mr. Sadtler. It discussed essentially the quantity of dye absorbed by quartz powder. For details the reader must be referred to the full paper, which will be published in the Transactions.

USE OF ULTRA-VIOLET LIGHT IN LABORATORY AND IN PRACTICE.

In the afternoon, Dr. CHARLES BASKERVILLE delivered a very interesting lecture at the College of the City of New York on the use of ultra-violet light in the laboratory and in practice. One very marked property of ultra-violet light is its power to transform O_2 into O_3 . This takes place best at low temperatures. It has been thought that the ultra-violet light from the sun forms O_3 in the upper atmosphere, which then settles down and oxidizes the impurities nearer the earth.

An electroscope is almost instantly discharged under the influence of ultra-violet light, but the action is quite slow if a quartz screen is interposed.

The most important application of ultra-violet light up to the present time has been in the testing of minerals. Certain minerals are inert, others effloresce, some phosphoresce, and still others both effloresce and phosphoresce.

Kunzite was discovered by aid of ultra-violet light. A large

number of spodumane samples were tested, and all found to be inert except one. Upon closer examination this one was found to contain a hitherto unknown mineral which was named Kunzite, in honor of the discoverer, Dr. George F. Kunz.

All the minerals from Mono Lake are phosphorescent after treatment with ultra-violet light. Some of the same minerals from other localities are not phosphorescent.

The genuineness of diamonds may be proven by their phosphorescence after being subjected to ultra-violet light.

The most important practical application at the present time is for testing the purity of tailings or precipitates, and for sorting minerals. A crushed mineral may readily be separated into an efflorescent and a non-efflorescent portion by sorting while acted upon by ultra-violet light.

The same principle is made use of by the New Jersey Zinc Co. for testing willemite concentrates and tailings. The willemite is efflorescent while the gangue is inert. If the tailings contain no efflorescent specks it shows that the concentrating tables are working well. The eye can also roughly judge the degree of concentration in the concentrates.

TUESDAY MEETING.

PYROMETERS.

The main part of the Tuesday meeting was devoted to an exhibition and discussion of various types of pyrometers. A brief introductory talk on this subject was made by Dr. E. F. ROEBER.

First, concerning the definition of temperature. Our senses tell us whether a substance or a body is hot or cold, but they do not give us a temperature scale. All measurements of the temperature of the body are essentially based on using a body of comparison, which is called a thermometer or pyrometer. We bring this in contact with the body, the temperature of which is to be measured, and we wait until thermal equilibrium is established. As thermometric substance we use a body which has some property which greatly changes when the body is cooled or heated. This variable property we employ for the temperature scale. Thus in a gas thermometer the absolute temperature is directly given by the volume.

It is important that thermal equilibrium is established between the substance under test, and the thermometer before a reading is taken. This is especially of importance for recording instruments for use in cases where the temperature changes very quickly. A sensitive thermo-electric pyrometer, for instance, will follow immediately all changes of temperature in air during a thunderstorm. It will show a sudden drop of temperature, for instance, while a mercury thermometer shows at the same time a less sudden drop, simply because it is not able to get immediately into equilibrium with the surrounding air.

The use of a thermometer for measuring temperatures makes us dependent on the substance of which the thermometer consists, mercury, gas, etc. We may choose two fixed temperature points. Then different types of thermometers, when properly calibrated, will give correctly the temperature at these two fixed points, but there will be variations at other points, though they may be small. Now, there is really no reason why one substance should be better than another substance, if we had not the fact that the laws which govern what we call perfect gases are so simple and uniform that we conclude that these regularities are based on a simple and uniform constitution of gases, and the temperature uniformly indicated by them we define as *the* temperature. This is the reason why the gas-thermometer temperature is the standard.

But, really even those gases which are almost "perfect" show slight differences, and if we want to take those into consideration then we are in the air with our definition of temperature, and we cannot get on solid ground except we get free altogether from an arbitrary thermometric substance. We need a temperature scale which is absolutely independent of any special substance.

This can be accomplished in no other way but by using the second principle of thermodynamics, in its most general form, where it does not depend on any special substance. This leads to the thermodynamic temperature scale of Lord Kelvin. How it is possible to realize experimentally this thermodynamical temperature scale—in other words how it is possible to apply the corrections to the indications of our gas thermometers—is shown in detail in Planck's well-known book on thermodynamics. While we have theoretically a possibility of establishing an exact absolute temperature scale, it has not yet been established.

While this shows how far we are still backwards in an exact definition of temperature, enormous progress has been made in recent years in practice in the invention and application of thermometers and pyrometers. We have now decidedly what we may call a pyrometer boom, and while the progress has been rapid, there is no doubt that it will be even more rapid in future, since the absolute necessity of controlling temperature in all kinds of industrial processes finds due recognition only just now. For all practical purposes the gas-thermometer scale is perfect, and by comparison with its scale we can calibrate any other pyrometer. This is all that we need in practice. For practical purposes we need some fixed points, and we have them in the fusing points of pure metals. But it is evident that there should be an international agreement concerning the figures for these fixed points. The author referred to some cases in which the figures of the Bureau of Standards in Washington and by the German Reichsanstalt agree with each other, while different values are used in France. This must, of course, lead to confusion.

We know that in many industrial, chemical and metallurgical processes the maintenance of the correct temperature is absolutely essential. Formerly the manufacturers depended on the trained eye of the workman. The disadvantages are evident. All the knowledge is stored in the workingman. When he leaves, the knowledge how to regulate the temperature leaves with him. On the other hand, if we use a pyrometer, we need to establish once for all the correct series of temperatures required for economical working and make a record, and we can then duplicate this record all the time. The pyrometer makes us independent of the personal equation of the furnaceman.

Further, recording pyrometers give us a method of control of the workman. We see from the record how efficient the different men are.

Moreover, if properly put up to the workingman he will look at a pyrometer, not as a spy, but as a help. It gives him some guide to rely upon. Mr. Whipple, of the Cambridge Scientific Instrument Co., has proposed to establish a bonus system. A certain bonus should be paid to the workingman if he keeps all the time within certain limits of temperature.

This is not only so with pyrometers. The same experience has been made with CO₂ recorders; for instance, in the works of the New Jersey Zinc Co., or in the power plant of the New York Subway. In both cases the use of these recorders has greatly enhanced the efficiency.

BRISTOL THERMO-ELECTRIC PYROMETER.

Mr. F. F. SCHUETZ then presented a paper on the thermo-electric pyrometer of Prof. William H. Bristol, which has already been described in detail in our March issue (our Vol. IV., p. 115). The paper was fully illustrated by demonstrations of the instrument, various types of which were exhibited. A new type shown on this occasion is Bristol's new recording pyrometer, illustrated in Figs. 1 and 2.

It is well known that the actuating current for the indicating arm derived from a thermo-electric couple is so small as to shut out all direct inking or any continuous direct contact of recording arm with chart. Prof. Bristol employs a specially prepared chart which is rotated by a suitable clock movement, and an electrical measuring instrument controlling a light recording arm provided with a marking point. The record chart

is supported only over a portion thereof, the remaining portion over which the recording arm acts being unsupported and perfectly free to vibrate.

Vibration is periodically communicated to the unsupported portion of the chart by a tapping apparatus controlled by the clock movement. The marking point is normally free of this chart, and comes in contact with it only during the period of

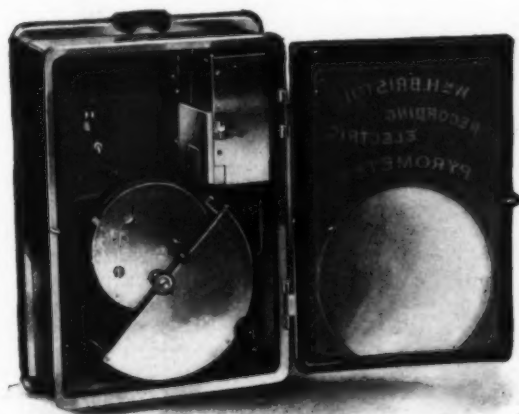


FIG. 1.—RECORDING PYROMETER.

vibration. Because of the delicacy of the recording arm no attempt is made to ink with it, but through the vibration of the chart vibration is communicated to it so that the impact is gradually taken up by the recording elements and the pointed end of this recording arm caused to touch the chart.

The chart itself is of the circular disc type, 8 inches diameter, such as used in the usual recording voltmeters, ammeters, pressure gauges, etc., and is semi-transparently smoked over

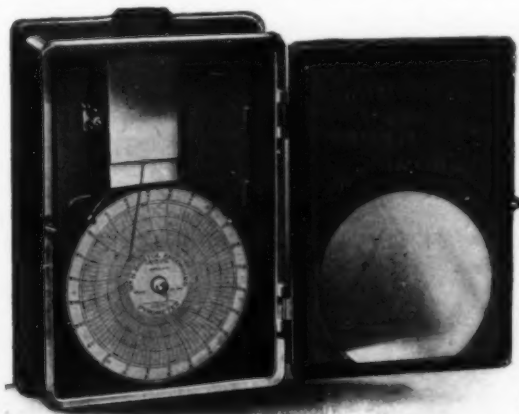


FIG. 2.—RECORDING PYROMETER.

its active surface so as to only partially obscure the graduations. The surface of the chart is so sensitive that it may be marked by a hair, and as the pointed end of the recording arm touches it a small amount of the soot is removed, leaving a small white dot visible. The frequency of vibration is such that a continuous line is marked upon the chart by the recording arm. There is, therefore, but little strain placed on the arm, and practically no friction between arm and chart to affect the sensitiveness of the movement of said arm, the con-

tact being only momentary. After the record is complete, the chart is removed and fixed by dipping into a fixative solution. This permanently fixes the record so that the chart may be filed away.

The second part of Mr. Schuetz's paper dealt with the application of pyrometers in commercial work. The author pointed out that with the availability of a comparatively inexpensive and practical pyrometer, capable of standing rough usage, pyrometers may now be used for such work for which they were not employed before. Experience has shown that the workmen take a positive interest in the pyrometer, and

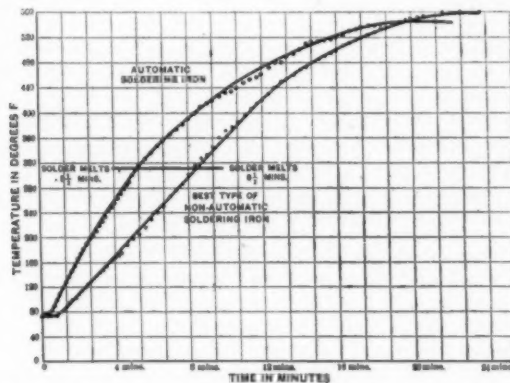


FIG. 3.—COMPARATIVE TESTS OF ELECTRIC SOLDERING IRONS.

that whenever such instruments are introduced the efficiency of operation is improved.

For the instantaneous determination of the temperature of molten metals, Prof Bristol employs an interesting modification of the old thermo-couple, the wires at the hot end of the couple being left separated. The couple is dipped into the bath, which makes an electrical contact with the two elements,

completing the circuit and immediately indicating the temperature by a suitable indicating device. When the couple has worn away to a length of about 1 foot the fire end is replaced by a new one.

This form of couple with pointed ends also affords a most convenient means for determining the temperature of metallic plates, and has been used to determine the proper temperature of the plates used in the manufacture of brazing powder and of zinc plates in the rolling of zinc sheets.

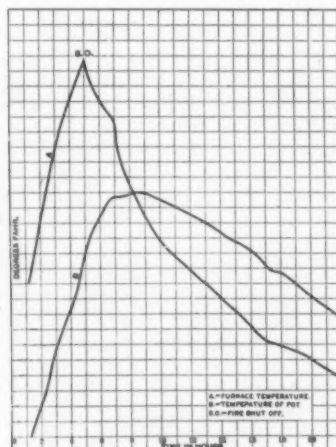


FIG. 4.—ANNEALING FURNACE RECORD.

This form of couple is sometimes provided with a small thin contact plate, especially when used to determine the temperature of electrically non-conductive objects.

Of the many practical applications mentioned in the paper we may mention only the application to comparative tests of electric soldering irons. The two curves in Fig. 3 represent the results obtained with an automatic iron and with a non-automatic iron, with the time plotted as abscissa and the temperature as ordinate.

Readings were taken about every $7\frac{1}{2}$ seconds; and it will be seen that it required the automatic soldering iron but $5\frac{1}{2}$ minutes to reach the soldering temperature, whereas the non-automatic soldering iron required $8\frac{1}{2}$ minutes.

Another interesting application is in connection with lead-hardening baths. The use of pyrometers has here resulted not only in more uniform results, but in a longer life of the pots containing the lead. The two curves A and B in Fig. 4 show respectively the fire temperature and the temperature of the box containing the articles to be annealed. It will be noted that the temperature of the box rose for some time after the fire was shut down.

In the handling of many small parts, such as in the manufacture of watches, clocks, etc., a very convenient device has been constructed by a large clock company. The pot containing the articles is adapted to the hot end of the couple, and the couple used as a handle for inserting the pot into the furnace or into an ordinary forge fire. By revolving the pot it is heated perfectly uniformly and is removed as soon as the required temperature is reached, as shown in the indicating instrument, all guess work being thus eliminated.

HERAEUS-LE CHATELIER PYROMETER.

Dr. RICHARD MOLDENKE, Watchung, N. J., the genial secretary of the American Foundrymen's Association, then presented a paper on the well-known thermo-couple of Le Chatelier, as constructed by Heraeus. As is well known, the combination of pure platinum with an alloy of platinum with 10 per cent rhodium is employed in the same.

When the Le Chatelier pyrometer was first brought to the attention of the scientific public, and its excellence was realized, the German Government, through the Physikalische Reichsanstalt, undertook to determine the proper length, thickness and resistance of the couple which would give the best results, and could therefore be made standard. This was found to be a length of 58 inches, a diameter of the wire of No. 23 B. & S. gauge, and a resistance of 1.6 ohms. To get as near perfection in the preparation of the element as possible, the work of preparation was entrusted to Dr. W. C. Heraeus, the proprietor of the platinum works of that name in Hanau, Germany.

Dr. Heraeus, in working out the problem of supplying a couple which would be absolutely uniform and interchangeable at all times, provided two blocks of the necessary materials worked up to uniformity in structure and perfectly pure. Thus there is available a supply that will last far beyond a lifetime, and the first as well as the last couple is absolutely the same. This feature alone makes the pyrometer a true standard.

After describing briefly the principle of the instrument the author referred to his personal experience with the instrument in daily use in malleable cast iron foundries, where the temperature of the annealing ovens has to be taken regularly, and where a small fortune is at stake continually. Perfect results were obtained by an ordinary laborer, selected for trustworthiness, recording the temperatures observed for the benefit of the annealer in charge.

The ease with which the instrument is applied in the particular case in point is shown by the fact that where formerly the Siemens water pyrometer occupied the attention of two men in 24 hours, with the La Chatelier pyrometer, the galvanometer located centrally, and the ovens fired up, the thermometer is introduced into the oven, and while the man walks to the galvanometer the pointer has reached the stationary point, the reading is taken, the thermo-couple moved to the next oven, and so on. A few hours in the twenty-four is all that is required, and the results are accurate easily within 25° F. Parenthetically it may be said that the temperature of the coldest pot of the annealing oven for annealing malleable castings should not fall below $1,250^{\circ}$ F., nor go above $1,350^{\circ}$ F., this for the best results with fairly heavy castings and on furnace iron. What sins have been committed before a good pyrometer was available can be better imagined than described.

For taking the temperature of molten metals, a clay tip arrangement is essential, since this can be dipped into the ladles directly without injury and the temperature observed within half a minute or so. The author then referred to various uses of this pyrometer, for instance, in connection with open-hearth glass works, chemical works, potteries, and especially in steel works.

Even the boiler room can come in for a share of attention. The author referred to a case in which he had to solve the smoke problem in a plant in Pittsburg, and only after applying the Le Chatelier pyrometer to every possible place in the 1,800-hp. installation was the solution found. The breeching and base of the stacks gave entirely too high temperatures for proper economy. An investigation of the grates showed that they were burned out too often. Smoke poured from the stacks of the over-worked boilers, and these had to be kept going at full pressure night and day. The stacks were raised an extra 50 feet one by one, making 150 feet in height each. The coal now danced on the grates. Plenty of oxygen got at the fuel. The grates did not burn out after this, and the boilers gave full value without a particle of smoke.

Here was a case where the best of stokers, a coal-conveying system and ash removal were installed with the highest class of water-tube boilers, and yet the overcrowding which always takes place in a live growing works brought about a lot of trouble quickly solved when the pyrometer was applied. There are others who might profit by doing this at their plants.

In conclusion, Dr. Moldenke, who was a member of the Jury of Awards at the St. Louis Exposition, remarked that while Mr. Engelhard, as the American representative of the Heraeus-Le Chatelier pyrometer, received the gold medal for his exhibition, they would have given the grand prize to Prof. Le Chatelier had he made a personal exhibit.

A Heraeus-Le Chatelier pyrometer had been placed on exhibition by Mr. Charles Engelhard at the meeting.

RESISTANCE AND RADIATION PYROMETERS.

Mr. R. S. WHIPPLE, of the Cambridge Scientific Instrument Co., of Cambridge, England, who has distinguished himself in developing methods and uses of pyrometry, made a very interesting speech. He thought the little compensator in the Bristol instrument, by means of which changes in the temperature of the cold end are compensated, was quite ingenious. He agreed that it is necessary to pay attention to constancy of the temperature of the cold end. He has always recommended the use of some waste steam, which is available in every plant. With the careless methods often used at present in many plants considerable errors may arise, due to changes of temperature of the cold joint.

Mr. Whipple congratulated American manufacturers and engineers on their readiness to adopt pyrometers in their plants. When the use of a new thing is recommended to an English manufacturer, he is liable to ask whether his competitors are using it or not. The American manufacturer does not ask such a question, but simply wants to know what advantages the instrument would bring him.

Mr. Whipple then described three instruments made by the Cambridge Scientific Co. and placed on exhibition. The first was a Féry radiation pyrometer, a detailed illustrated description of which was given in our Vol. III., page 478, exhibited by the Wilson-Macaulen Co., of New York City. The second and third were electric resistance pyrometers.

Resistance pyrometers have a wide range, from the lowest temperatures to $1,200^{\circ}$ C., or nearly $2,200^{\circ}$ F. They can be placed in inaccessible positions and read from a considerable distance. They are extremely sensitive and respond quickly to changes of temperature. In conjunction with the Whipple temperature indicator they give direct readings of temperature requiring no correction. In conjunction with the Callendar electric recorder they furnish a continuous and visible record of temperature, extending over a day or week, and the

temperature can be read off at any time without disturbing the recorder.

Fig. 5 shows the resistance pyrometer connected to a Whipple temperature indicator. By depressing the contact key F and by turning the middle head H, the resistance of the indicator is made to equal that of the thermometer, the galvanometer needle at B showing when balance is obtained. The temperature in degrees is read off directly on the scale at A.

The indicator need not be near the thermometer, but may be at a considerable distance from it without effecting the accuracy of the readings. The instrument is so constructed that rapidly varying temperatures may be readily followed and measured. It is easily portable, and by means of the instru-



FIG. 5.—RESISTANCE THERMOMETER WITH WHIPPLE INDICATOR.

ment any workman without any electrical knowledge may find the required temperature in 2 or 3 minutes. No corrections of any kind have to be applied.

Mr. Whipple then showed several interesting lantern slides from pyrometer practice. One of these is reproduced in Fig. 6, which is an annealing-furnace record, obtained with a resistance thermometer in combination with a Callendar recorder. Such a record discloses at a glance to what extent the temperature of the furnace has deviated in either direction from its proper value. Incidentally it shows quite clearly how often the fires have been fed, each firing producing a transient heating effect, which is shown by a well marked peak on the record. The temperature is about 700° C., and the portion

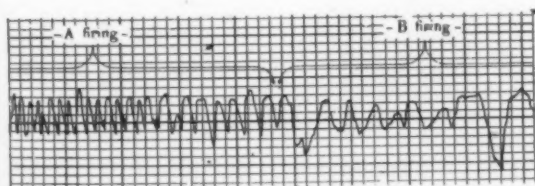


FIG. 6.—ANNEALING FURNACE.

of the record shows an extension over about 9 hours. Each space on the vertical scale corresponds to 4° C.

It is interesting to note the difference in the firing of the two men A and B. One of the two firemen was a young man, and his nervous state of mind is shown by the way how often

he fired. The other was an old man, and took things far more quietly. The record also shows distinctly when he took his little nap.

BROWN PYROMETERS BASED ON DIFFERENCE OF EXPANSION.

MR. RICHARD P. BROWN, of Edward Brown & Son, of Philadelphia, Pa., then exhibited and described two pyrometers based essentially on the difference of expansion of two different materials. This is, of course, the underlying principle of the ordinary mercury thermometer, the action of which is based on the fact that mercury expands very much more under the action of heat than the glass tube in which it is contained and the scale with which it is combined. The same principle is made use of in quite an interesting way in the two Brown instruments.

The first is a recording pyrometer, which is a modification of the Brown hot-blast pyrometer, which was originally designed for use on the hot-blast main-of-blast furnaces, but has now come into general and extensive use for a great variety of operations for measuring temperatures up to 1,500° F. or 800° C. The Brown recording pyrometer keeps a record of the temperature on a chart for 24 hours.

Inside a steel tube are graphite rods, having practically no expansion, and the indication of the instrument is caused by the difference in expansion of the outer seamless drawn-steel tubing and the graphite rods, 12 inches long. Above the graphite rods is steel tubing with pieces of brass or other metals, by which the instrument is accurately compensated for various insertions in the heat. No matter whether the stem is inserted 12 inches or 24 inches in the heat the instrument will indicate accurately. In the case of head of the instrument is a multiplying movement, which multiplies the slight difference in expansion, and an adjustment is furnished in the movement by which each instrument may be adjusted accurately. Also an adjustment is furnished on the outside of the case by which the pyrometer is adjusted to the temperature of the atmosphere before use.

The second pyrometer exhibited by Mr. Brown was a "quick-acting platinum pyrometer," which has a thin strip of platinum 8 inches long, hung between heavy iron frames. When the end of the stem of the instrument is inserted in the heat the thin platinum strip is directly connected to a multiplying movement, which moves the pointer. When the thin platinum strip is heated the pointer quickly moves around the dial, and when the platinum has ceased to expand the pointer stops, and as the iron frame begins to heat up the pointer will begin to move back again. The maximum temperature indicated is the temperature of the furnace or heated space. As soon as the temperature is noted the pyrometer is withdrawn and can be used again when the instrument returns to the atmospheric temperature, which will be in about 15 minutes.

As the time necessary for the platinum to heat up will be only about 10 or 15 seconds, the iron frame will not have risen about 200°, probably less, and as the platinum will only be subjected to the high temperature for so short a time in each test, the indications of the instrument will not be altered, due to any deterioration. This instrument can be used for measuring temperatures up to the melting point of platinum.

PRICE THERMO-ELECTRIC PYROMETER.

A Price thermo-electric pyrometer made by the Electric Dental Specialty Co., of Cleveland, Ohio, was then exhibited. Two couples were shown in connection with the instrument for demonstration purposes, but it was explained that out of the some 125 different industries which are now using this type of pyrometer very few have the same kind of couples, since the material of the wires and also their length and size in the couples depend upon the conditions of the several industries in which the instrument is to be used. For high temperatures platinum is required in order to get reliable results, but for low temperatures other metals are equally good.

The two wires of the couple are insulated from one another

in several different ways. Where compactness is of importance they may be run through magnesia tubes $\frac{1}{8}$ inch outside diameter and having an 18 B. & S. hole. Another way is to secure clay tubes sufficiently refractory to stand high temperatures having holes pierced through from end to end. Sometimes the wires may be held apart by asbestos cord woven in and out; asbestos tubing is excellent where there is a great deal of vibration.

For higher temperatures which reach about 1,600° C. or more it is necessary to use a refractory porcelain or semi-clay porcelain tube, which is closed at one end, unless very sudden changes of temperature are to be recorded, when the end of the couple or junction is run through outside. It is still better if these tubes are glazed on the outside to prevent any possibility of contamination to the thermo-couple by furnace gases.

Where the heat is extreme these tubes are often again supported by large fire-clay tubes having holes straight through, allowing the end of the former tube to project through a short distance. This prevents the sagging of the porcelain tube when it is in a horizontal position, and also prevents it from breaking. When inserted in a clay tube it has been found to last three or four times as long.

When working at higher temperatures with platinum, iridium and rhodium, it has been found better to use two couples in series, in order to get greater precision or accuracy of temperature. In this case the cold end of the couple need not extend far from the heat but may be joined there by certain wires of a metal, which develop a counter-electromotive force equal to that set up by the cold junction. This has been perfected to a high state of efficiency, allowing, in



FIG. 7.—PRICE PYROMETER.

some cases, the cold end of the couple to be at temperatures as high as 500° C., thus greatly reducing the length of the couple. But at temperatures less than 850° or 900° there are several metals that may be used and can be relied upon. Brass and alloy of copper and nickel gives a very high and steady e. m. f. Iron used with the same alloy is almost as good unless it is exposed to injurious furnace gases.

The method used in the Price pyrometer for its indicating instrument is to suspend the armature on one single point, since when an armature is suspended between two points and turned over sometimes in a horizontal position, it is bound to cause a considerable amount of friction, which consequently prevents accurate reading of temperatures.

The incoming leads to the armature are crimped suspension springs extending 4 or 5 inches above the armature. Where they are attached the inlets to the armature are very close together, but the ends at the top are further apart to give the proper torsion and resistance to the current. The crimp in these springs allows a lengthwise spring of nearly $\frac{1}{4}$ inch in that distance, so that it is almost impossible to break them.

When the armature is suspended in its place above the point a large proportion of its weight is carried by these suspension springs, and only enough left on the point to keep it centered around the core and in the field. This reduces its internal friction to almost nothing, making it possible to get that distance, so that it is almost impossible to break them.

The thermo-electric pyrometer can readily be arranged to give a continuous record of temperature. At every minute a tiny spark leaps from the indicating needle and passes through a continuous sheet of paper to a contact below. This paper has marked upon it the hours and parallel lines representing the temperature. It is just as easy to see the temperature on this recording instrument as on the ordinary indicating instrument, as the paper simply passes underneath the needle, leaving the ordinary scale in view just the same as usual. It has the great advantage that it has no inclination whatever to retard the needle but allows it to show the exact measure of the temperature both on the scale and the hole through the paper.



FIG. 8.—SWITCHBOARD.

These holes appear similar to the puncture of a pin. One of these recording instruments may be in a head office, and it may be a single, double or triple machine, recording the temperature of one, two or three places at once, while at the plant, for the use of the fireman, is an indicating instrument connected to the same couple or couples as that used by the recording instrument, so that the fireman always sees just the same temperatures as the recorder is continuously marking. The wire connecting the couple with the galvanometer may even be a mile in length.

For the calibration of these instruments fixed points are, of course, required. Copper melted in a graphite crucible, with its surface protected with powdered graphite, gives a very definite and sharp fixed point at 1,083° C. when freezing. (Day and Allen's value is 1083.6 in a reducing atmosphere.) During the freezing period the temperature remains quite constant, giving ample time for observations. Certain grades of gold in a very loose form or rolled exceedingly thin and bunched up in a small bunch will usually melt within 5°.

The instrument is very well adapted for immersion in small specimens of steel and other metals for the study of the cooling curves and in connection with annealing, hardening, etc.

While thermo-electric pyrometers were formerly used almost exclusively for high temperatures only, the Price pyrometer is now also made to give temperatures below as well as above the freezing point of water. When the outside junction is carefully wrapped in wool of sufficient thickness, in order to maintain the temperature uniform (wool being supposed to be the best non-conductor of heat that we have, being five times as efficient as asbestos), a very reliable instrument can be made to show the temperature from 50° below zero to the boiling point of water.

Fig. 7 shows a Price pyrometer. In this illustration the connections marked "Fig. 1" show how the couples are joined together at their extremities, their separate ends leading to the meter, while "Fig. 2," in the same illustration, represents the appearance of a couple, such as is used in tempering vats, compressed steel chambers, glass and steel annealing ovens and many such places not requiring above 850° C.

Fig. 8 shows a switchboard used in connection with the Price pyrometer in case a larger number of couples are to be

connected to the same instrument, so that by means of one indicating instrument the temperature in different furnaces may be measured. The contact points of this switchboard cover both sides of the switch, similar to a knife switch. By moving the switch to the contact point, which is connected to any special furnace, the temperature of this furnace will be indicated. As many as forty or fifty couples may be connected in this way to the same outfit. "The positive wires of all the couples are connected together by one leading to the black screw at the top of the switchboard, while the negative wires of all the couples lead from the lower binding post of the instrument.

INSTRUMENTS WITH RADIATION AND RESISTANCE PYROMETERS.

Last, not least, some experiments were made by Mr. C. H. WILSON, of the Wilson-Maculien Co., immediately after the meeting. As was mentioned above a type of the Féry radiation pyrometer was exhibited by this company. In this connection it is interesting to state that the Cambridge Scientific Instrument Co. has now succeeded in making this instrument recording. As far as we know this is the first and only type of radiation pyrometers which has been rendered a recording instrument. Radiation and optical pyrometers are specially useful, or rather, absolutely necessary, in all cases in which it is necessary to measure the temperature from a distance. With optical pyrometers the measurement is restricted to a wave length in the visible spectrum, while the Féry pyrometer is based on the total radiation in connection with the Stefan-Boltzmann law. For description of the Féry pyrometer reference must be made to our Vol. III., page 478.

Mr. C. H. Wilson exhibited the Féry radiation pyrometer in practical use and showed how easily it is handled. He also made a very interesting experiment on the determination of the temperature of a gas-heated muffle furnace, both with a Féry radiation pyrometer and with an electric resistance pyrometer. It was shown that for temperatures up to that for which the resistance pyrometer can be used, both instruments, although based on entirely different principles, give the same results.

PRIMARY CELL.

A paper on the Decker primary cell was then presented by Prof. FRANCIS B. CROCKER. There is nothing new of an electrochemical nature in it. The whole novelty rests in the method of mechanical construction, but Prof. Crocker pointed out that just in this point all former primary cells were found wanting.

The cell of Mr. F. A. Decker, of Philadelphia, is of the double-fluid type, with zinc plates in dilute sulphuric acid and graphite plates in a solution of sodium bichromate and sulphuric acid. Each zinc plate and the dilute sulphuric acid surrounding it are contained in a flat porous cup. In order to form such a cup two unglazed earthenware plates with thickened edges and diagonal strengthening ribs are shaped separately in steel moulds, a special clay mixture being employed to obtain any desired porosity. These plates are made extra thick to prevent warping in burning and produce true flat surfaces and straight edges. They are then united to form a flat cup, after which it is ground down on each surface to the desired thinness. In point of fact the finished walls are very thin, so that light will show through them, which enables the degree and uniformity of thinness to be readily determined. The grinding operation involves little time or skill, being performed by half-grown boys. In this way a flat porous cup is obtained, perfectly true on all sides and with walls much thinner than could possibly be made if it were attempted to form a complete cup in the first place. The graphite plates are placed directly against the outside walls of the porous cups. The thinness of the porous wall and the nearness of the zinc and graphite plates results in a low internal resistance of the cell.

The containing vessels are made of hard rubber, and special

provision is made for the supply of the electrolyte to the cells through pipes, etc. A whole battery consisting of several cells is so assembled that with all its cells, conduits and porous cups, it constitutes one substantially integral mass with no loose joints or parts. For details of construction, the reader may be referred to the complete article of Prof. Crocker, in *Electrical World*, Oct. 13.

In tests made by Prof. Crocker, in which the cell was discharged from 1.9 to 1.3 volts, an output of 14.7 watt-hours per pound of total weight was found. A cell weighing 17 pounds gives 150 amp-hours at an average of about 1.7 volts, or about 250 watt-hours, which is equivalent to one-third of a horse-power-hour.

The weight of zinc, sulphuric acid and sodium bichromate required to give 1 hp-hour in the Decker cell, assuming all materials to be thrown away after being used once, cost about 35 cents. The corresponding cost for the Lalande cell is \$5.30. Since the Lalande cell has numerous applications, it is thought that the Decker cell will have an even broader field of usefulness, especially for automobiles and for lighting railway trains.

The paper was briefly discussed by Mr. E. A. Sperry, Prof. Tucker and Mr. Carl Hering.

After, at motion of Mr. Morehead, thanks had been expressed to Columbia University and all others who had shown courtesies to the Society during the convention, the meeting adjourned.

EXCURSIONS AND SOCIAL FUNCTIONS.

Prof. S. A. Tucker, of Columbia, courteously showed to the visiting members the finely equipped laboratory of applied electrochemistry in his charge, in the basement of Havemeyer Hall. A full description of the laboratory was given in our May issue (Vol. IV, p. 175).

On Monday afternoon, after Dr. Baskerville's lecture, the members visited the Waterside station of the Edison Company, under the auspices of Dr. Elliott. The visitors saw the operation of the plant at its busiest time, namely, just when the evening lighting load was coming on.

On Monday evening a subscription dinner was served in the hall of the Liederkranz Society. Though the attendance was small—below forty—the affair was extremely enjoyable. Mr. H. B. Coho again distinguished himself as toastmaster.

A most interesting excursion was made on Tuesday afternoon to the large electrolytic copper refinery of the American Smelting & Refining Co., at Maurer, near Perth Amboy, N. J.

Many members attended the dinner and the reception given to Sir William Perkin, on the evenings of Saturday and Tuesday, respectively.

Electric Smelting of Iron Ore.

In the development of an iron industry California, like Canada, has been handicapped in the past by the lack of coke; and the same trend of ideas which led the Canadian Government to a systematic and careful study of electric smelting of iron ore has induced progressive Californians to try electric smelting. Water-powers for cheap generation of electric power are abundant in California as in Canada.

This interesting work is to be carried out by the Northern California Power Company, the president of which, Mr. H. H. Noble, takes very great interest in this development, in conjunction with the Shasta Iron Company.

The Héroult process will be used and the work will be conducted in a general way along the lines of the celebrated Soo experiments.

Mr. R. A. Turnbull, who participated in the smelting experiments at Sault Ste. Marie, and who is now Dr. Héroult's representative in Canada, is at present in charge of the engineering side of the undertaking. M. Petinot, who has been connected with Dr. Héroult's electric smelting plants in Europe, will be in charge of the work at the spot.

The Extraction of Metallic Sodium.

By C. F. CARRIER, JR.

A brief consideration of the purpose and scope of this article must be its own excuse for being. While the literature of sodium is quite abundant, and even extended treatises have been devoted to this single topic, there is no one publication of any kind in which the bibliography has been given with any degree of completeness. It is hoped to present, in systematic form, a bibliography of the subject more complete than has previously been published, together with brief descriptions, which will make it a history of the art.

The classification used is based on the Dewey Decimal System, as elaborated by Mr. Adolph L. Vöge, of the Concilium Bibliographicum, of Zurich, Switzerland. For the sake of completeness, all processes will be mentioned, whether they have had successful technical application or not, but the relative importance will be to some extent shown by the space devoted to the description, in absence of some specific comment on that phase of the matter.

SCHEME OF CLASSIFICATION.

669. Metallurgy.
669.733. Sodium.
669.733.(09) History of Sodium.
669.733.36 Sodium by Chemical Processes.
.362 Reduction of Sodium Carbonate.
.363 Reduction of Sodium Hydroxide.
.364 Reduction by Carbides.
669.733.372 Electrothermic Processes.
669.733.374 Electrolytic Fusion Processes.
.374.2 Electrolysis of Fused Salt.
.374.3 Electrolysis of Fused Carbonate.
.374.4 Electrolysis of Fused Hydroxide.
.374.5 Electrolysis of Fused Nitrate.
.374.6 Compound Electrolysis.
669.733.375 Electrolytic Solution Processes.
.375.2 Using Sodium Amalgam.
.375.3 Using Organic Solvents.
669.733.6 Preparation of Sodium Alloys.
669.733.9 Uses of Sodium.

References:

- Proc. Am. El.-ch. Soc., 8, 161 (1905).
This Journal, 3, 107 (1905).

HISTORY, 669.733 (09).

Electrochemistry was very essentially concerned in the discovery of metallic sodium by Sir Humphrey Davy, who subjected the fused hydroxide to electrolysis, but it was over eighty years before any successful technical application of this discovery was made. While his work was successful in this qualitative way, nothing much seems to have been accomplished in fixing conditions required, or in making clear the theory of the dissociation. Davy supposed that if the caustic soda were anhydrous, no hydrogen would be evolved at the cathode. It is also worthy of note that he made use of the principle of using the current to heat the bath as well as accomplish the electrolysis. The eighty years following were taken up with chemical processes and a few spasmodic attempts at electrochemical processes.

References:

- Phil. Trans. (1808); 1 and 133.
(1809); 39.
(1810); 16.
Ostwald's "Klassiker," No. 45; 55.
"History of the Alkali Metal Industry," Lumiere Electr., 44; 179 (1892).

NOTE:—The number of the volume to which a reference is made precedes the ";", the page follows the ":", and the year is enclosed in the ().

CHEMICAL PROCESSES, 669.733.36.

It would be interesting, as well as instructive, to make a detailed study of the strictly chemical processes for the extraction of sodium, but owing to the vast quantity of literature pertaining to this branch, only a brief outline can be given, sufficient merely to show their relation to processes involving the thermic action of the electric current.

Gay-Lussac and Thenard reduced sodium hydroxide in an iron tube by treating with iron filings at a white heat, but at the suggestion of Curadon, carbon was used in place of iron as the reducing agent. Various improvements in the details of the process were made by Brunner, Wohler, Donny, Mitscherlich and others, but the metal was first prepared on a large scale by Deville. By mixing chalk with the charge he kept it loose and open, thus facilitating the reduction. If this is not done, the carbonate or hydroxide will melt and collect in the bottom of the retorts. The sodium thus produced was used in the extraction of aluminium. Castner improved upon this by using a "carbide of iron" as the reducing agent. This carbide was prepared by heating a mixture of tar and iron filings to a high temperature in a closed vessel. The reduction can be carried out at a lower temperature than when carbon is used, and thus there is a saving of fuel, and still more important, the life of the retorts is greatly lengthened. Several other inventors were working on the chemical processes at about this same time. O. M. Thowless melted the alkali compound to be reduced, and then added it gradually to a mass of incandescent carbon. In this way he claims to avoid the formation of certain undesirable compounds in the retort. H. S. Blackmore heated a charge made up of calcium hydroxide, ferric oxide, sodium carbonate and carbon. In the process of Netto the temperature is kept lower than in the other processes, so that only the hydroxide will be reduced, the carbonate formed during the process being allowed to collect in the bottom of the retort and drawn off from time to time.

References:

- Thorpe, "Dictionary of Applied Chemistry," 3, 422.
Castner, U. S. P., No. 342,897, of June 1, 1886.
Thowless, U. S. P., No. 380,775, of April 10, 1888; U. S. P., No. 380,776, of April 10, 1888.
Blackmore, U. S. P., No. 391,110, of Oct. 16, 1888.
Netto, U. S. P., No. 460,985, of Oct. 13, 1891.

There are also several chemical processes that are indirectly related to electrochemistry in that they use electrochemical products as reducing agents, or depend upon some hypothetical action of electrical energy.

The process of Hasenclover & Sons utilizes static discharges to "increase the activity of reducing material." No data are available showing the efficacy of this expedient. Ger. P. No. 65,921 of Jan. 11, 1891.

The use of calcium carbide for the reduction of melted sodium hydroxide was patented by Wolfram in 1898, and the reaction has been studied by Kügelgen. The products of the reaction are calcium oxide, sodium, hydrogen, carbon monoxide and a little calcium carbonate. Kügelgen also found that fused sodium chloride was also partially reduced by calcium carbide, but in both cases the yield is very poor.

References:

- Wolfram, Ger. P. No. 101,374 of March 23, 1898 (also Eng. P.).
Kügelgen, Zeit. f. El.-vm. 7; 576 (1901).
For the preparation of chemically pure sodium, Beketoff and Scherbatscheff reduce sodium aluminate with magnesium and distill off the sodium. Bulletin of the Academy. St. Petersburg (1894).

A modification of the Wolfram process invented by Moeser and Eidmann has been assigned to the Chemische Fabrik Greisheim Electron. This consists in reducing the fluoride by means of calcium carbide instead of the hydroxide. It is especially recommended for potassium reduction. U. S. P. No. 710,493 of Oct. 7, 1902. Also patents in Ger., Fr., and Eng.

Another process acquired by the Chemische Fabrik Griesheim Electron is that of Specketer and Weber for reducing the alkali fluorides in a suitable still by means of aluminium. U. S. P. No. 730,979 of June 16, 1903. (Also patents in Ger., Fr., and Eng.).

With the introduction of the electrolytic processes for the preparation of aluminium, the main use of metallic sodium was taken away and the older chemical processes were abandoned, possibly never to be revived. Castner could not compete with the Hall process, and so attempted to discover a cheaper process of extracting sodium. This led to the successful electrolysis of fused sodium hydroxide, but did not reduce the cost of the metal enough to enable it to maintain its place as a reducing agent in the preparation of aluminium. Through force of necessity, new uses for sodium had to be found which were the beginnings of the modern sodium industry.

ELECTROTHERMIC PROCESSES, 669,733-372.

The great difficulty with the chemical processes was their low thermal efficiency and the tremendous wear and tear on the retorts. It would seem that the possibilities of efficient internal heating by means of electric energy would have enabled a modified chemical process to remain in the field, but

the problems of condensation and furnace construction give no present evidence of solution.

Probably the first suggestion of using electrical energy to supply the necessary heat for reducing sodium oxide by carbon was that of C. A. Faure in 1882. He placed the charge in a magnesia retort and passed the current through it, or through resistors in the walls of the retort, a current of nitrogen or hydrogen being used to carry the metallic vapor to the condenser. Eng. P. No. 6,058 of 1882 and No. 5,489 of 1883.

The first application for a patent in this country was that of

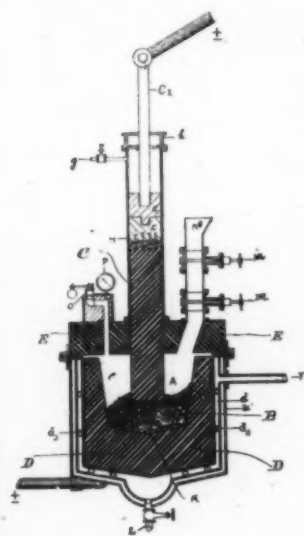


FIG. 1.—COWLES FURNACE.

Bradley and Crocker in 1885. They accomplished the heating by passing the current through the walls of the wrought-iron tube that formed the retort. There seems to be no self-evident cause for failure of this process, but it is possible that the tubes, being hotter than the charge, were very rapidly consumed in reducing the sodium, or the thermal efficiency may have been too low. U. S. P. No. 335,499 of Feb. 2, 1886. This Journal 3: 128 (1905).

Parker and Robinson make use of the arc as a source of heat, and in order to "distribute the heat," have the charge of sodium oxide and carbon resting upon a mass of molten metal, which acts as one pole of the arc. Eng. P. No. 11,707 of July 23, 1889.

The most extensive work on electrotthermic processes has been done by A. H. Cowles. His first patent was applied for in 1895, and was followed by several others involving details of the apparatus, a progressive study of which shows some of the difficulties involved in the electrotthermic distillation of sodium. The fundamental idea is to reduce sodium aluminate by carbon under the influence of electrotthermic heating, condense the sodium vapor and tap off the aluminium carbide which collects in the bottom of the furnace. By adding some

non-volatile metal, as iron, copper, etc., an alloy of aluminium may be obtained instead of the carbide. The special features of the furnace (Fig. 1) are the devices for making it possible to work above atmospheric pressure. The carbon electrode C is introduced into the furnace by means of an iron rod passing through a stuffing box. To prevent reducing the pressure when fresh material is added, two valves m m are placed between the hopper N' and the furnace chamber. The cover is made air tight by the ring of insulating material between it and the body of the furnace. A safety valve and pressure gauge control the pressure in the furnace chamber. The sodium vapor and the gases produced from the reaction are forced through the porous carbon walls of the chamber and the sodium is condensed in the hollow space between the outer walls, the furnace b being surrounded by water jackets to keep the metal walls cool enough to act as condensers. A slow current of neutral gas is forced into the furnace by the pipe g in order to protect the electrode as much as possible.

References:

U. S. P. No. 676,575-76-77 of June 18, 1901.

U. S. P. No. 673,761 of May 7, 1901; also Eng. patents.

I. L. Roberts supports the charge in his furnace by means of a grating of conductors that are heated by the passage of the current. The reduced material gathers in the chamber beneath the grating. U. S. P. No. 629,394 of July 25, 1899.

In a French patent, H. Becker proposes to reduce sodium manganate, chromate, tungstate, aluminates, etc., by carbon, condensing the sodium and tapping off the manganese, chromium or alloy which may be formed according to the charge. The great advantage claimed by the inventor is the simultaneous production of two valuable metals, that by the conditions of the process and their own specific properties are readily separated from one another. In his monograph, "Die Elektrometallurgie der Alkalimetalle," Becker makes no specific description of the kind of distillation furnace in which he proposes to carry out this process. He states that the process has been tried successfully on an experimental scale. On pages 109-111 of the above mentioned monograph the Cowles process is assigned to the year 1901, and the Becker process to 1899, but it is to be noted that the Cowles patents were applied for in 1895.

References:

Fr. P., No. 288,274, of April 27, 1899.

This Journal 1: 422 and 575 (1903).

To the best of the author's knowledge, none of the electrotthermic processes are in commercial operation at the present time. It is probable that simply as sodium producers the high temperature and the deterioration of the apparatus prevent competition with the more simple electrolysis of fused sodium hydroxide.

ELECTROLYTIC FUSION PROCESS, 669,733-374.

In addition to the work done on the electrolysis of the more common sodium salts in the fused condition, some research work has been recorded on the more expensive compounds. Faraday investigated the sulphate, carbonate, borate and phosphate. Brester worked on the sulphate. The cyanide was found by Linnemann to be an excellent raw material if the commercial side is ignored. Extensive researches by Hittorf on fused nitrate, chlorate, etc., were incidental to his monumental work on the migration of ions. Kleiner-Fiertz took out an English patent on the electrolysis of cryolite as a source of sodium, and further work on the same line was done by Houmpe in Italy.

References:

Faraday, Ostwald's "Klassiker," No. 86.

Hittorf, Pogg. Ann. d. Phys., 72; 481 (1847).

Linnemann, Jr., f. Prak. Chem. 73; 415 (1848).

Kleiner-Fiertz, Eng. P., No. 8, 531 of 1886.

Houmpe, Electricita (1889); 307.

Brester, Jahresbericht f. Ch. (1866); 84.

ELECTROLYSIS OF FUSED SODIUM CHLORIDE, 669.733.374.2.

The electrolysis of fused salt has so many inherent advantages and such brilliant commercial possibilities that it is not at all surprising to find that this class embraces by far the largest number of attempts to prepare metallic sodium. With an ore almost chemically pure, in almost unlimited quantities at a very low price, and which is a good conductor when melted, the problem looks alluring, but in spite of these advantages not a pound of sodium is commercially produced by direct electrolysis of fused salt.

The difficulties are numerous. The melting point of sodium chloride is above bright redness, and has been fixed by various authorities at 750° to 950° C. The most reliable figures are between 800° and 850°. It is, however, not an easy matter to prepare a bath of fused salt. The selection of materials for furnace construction is made difficult by the fact that substances which will resist the action of melted salt, such as fire-clay, magnesia, etc., are readily attacked by sodium, and those which will withstand sodium are useless against melted salt. Even iron is somewhat corroded by sodium at the melting point of salt, and it is totally unfit to use as the container for the electrolyte.

The above difficulties might be overcome by cooling the walls of the container and maintaining a protective coating of the electrolyte, but the behavior of the electrolyte itself is a far more serious matter. It has been thought by some that the sodium unites with sodium chloride to form a sub-chloride, but the existence of this compound has not been conclusively proven. The Lorenz metal fog theory accounts for the depolarization quite as satisfactorily, and the high temperature of the bath is certainly a condition favorable to the formation of a metallic suspension in the electrolyte. It is also possible that chlorine is to some extent soluble in fused salt. Regardless of its exact nature, however, it is certain that some depolarizing agent is formed at one or both of the electrodes and diffuses or circulates to the other electrode, thus reducing the efficiency. This problem has been quite thoroughly investigated by A. Fischer in the Aachen Laboratory, but the author neglected to give the essential data showing the results obtained by overcoming the unfavorable conditions which he himself had noted. *Zeit. f. El.-ch.*, 7: 347-354 (1900).

As early as 1844, a patent appeared on the extraction of metals from fused salts. The inventor, Napier, used a container of conducting material lined with a refractory material on the sides, which was a non-conductor, the bottom serving as the cathode. Inasmuch as the metal was to be collected on the bottom, it is not probable that the apparatus was intended for the separation of sodium, but it is worthy of mention as being the first application, or conception, of that principle, frequently made use of in the design of furnaces for the electrolysis of fused salts. Eng. P. No. 10,362 of 1844, and No. 684 of 1845.

The first patent for a process specifically intended for the separation of sodium was that of James Watt. His apparatus consisted of a covered iron pot in which was a partition extending part way to the bottom, an electrode being located in each of the compartments thus formed. The electrolyte was a fused halogen salt of the alkali or alkaline earth metals. As cathode material carbon was used and the anode was of gold. The sodium liberated at the cathode was removed from the cathode compartment by distillation. In the light of present knowledge, this apparatus is self-evidently impracticable, and was undoubtedly purely hypothetical. Eng. P. No. 13,755 of Sept. 25, 1851.

The process of B. J. Dickson may have been worked upon an experimental scale at least, for he made use of sodium hydroxide as well as sodium chloride in his patent specifications. It is to be noted, however, that the use of sodium hydroxide as a raw material for the production of sodium had been made the subject of a patent long before the Castner patents. Eng. P. No. 2,266, Aug. 13, 1866.

Jablockoff attempted to improve on the apparatus of his

predecessors by enclosing the two electrodes in tubes which might be made of clay, porcelain or metal. The products of the electrolysis were to be carried off by the tubes surrounding their respective electrode, the sodium being vaporized. An earthenware pot heated externally held the electrolyte. This apparatus did not overcome any of the difficulties which made the previous processes impossible, and introduced some complications of its own. *Dingler's Poly. Jr.*, 251: 422 (1884).

Hoepfner attempted to overcome the disturbing influence of the chlorine by using a heavy metal as a soluble anode. The metal was placed in the bottom of the melting pot and the electrolyte above it. Upon electrolysis, the chlorine would, of course, unite with the heavy metal, but he assumed that the chloride would remain in a layer at the bottom of the pot while the alkali metal was liberated at the cathode above. The special metal cited as an example was copper. This would dispose of the chlorine very nicely, if it were only possible to get around the fundamental principle of electrodeposition and separate sodium from a bath containing copper. Ger. P. No. 30,414 of March 21, 1884.

The pioneer American worker in this field was A. J. Rogers. His apparatus was an elaboration of the same general features as are involved in the apparatus of Watt, and was no more successful. The chief distinction was that Rogers introduced the salt into the electrolytic cell in the molten state. U. S. P. No. 296,357 of April 8, 1884.

The first real steps in advance were developed in the apparatus of J. Olmolt, although it must be said that, from the practical standpoint, he was not much more successful than those who had gone before. The improvements seem to be due to his having had a somewhat better understanding of the materials of construction which must be used in order to meet the special conditions involved. The most marked variation from previous furnaces is in the substitution of reverberatory heating in place of heating by conduction through the walls of the vessel containing the electrolyte.

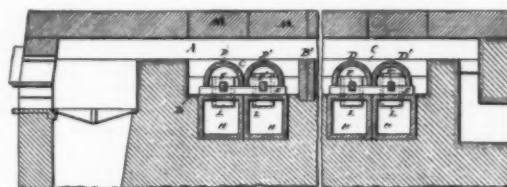


FIG. 2.—OLMOLT APPARATUS.

The hearth of the furnace (see Fig. 2) consists of a number of half retorts DD' arranged in pairs. These retorts resemble muffles with the flat side removed. The open side is placed downward and the molten electrolyte forms a liquid seal around the entire lower edge. The retorts are of fire-proof material and are lined with a material containing carbon. Each retort contains an electrode of carbon or a "suitable metal" FF'. The liquid seal prevents the furnace gases from coming in contact with the products of the electrolysis and also the products of the electrolysis from mixing with each other. The sodium is collected in the chamber H by passing from the retort D through the pipe L. It appears improbable that this furnace was ever given a practical trial, but a number of disadvantages can be pointed out. Electro-thermic heating is far more efficient than reverberatory heating that has to pass downward through the wall of a retort. The structural material problem has not been entirely solved, for there are some parts of the furnace that have got to withstand both salt and sodium. Maintenance charges would probably kill the process even if it were not a physical impossibility to separate sodium from melted salt with the electrodes in such close proximity to each other. U. S. P. No. 382,183 of May 1, 1888. Also patents in Ger., Eng., Fr. and Belg.

In 1886, F. Fischer described an apparatus that so closely resembles that of Watt (1851), although it did not have a gold anode, that he cannot be said to have profited by the experience of the intervening thirty-five years. Wagner-Fischer Jahresber (1886); 222.

Ludwig Grabau took out a series of patents between 1886 and 1890, covering improvements in both apparatus and electrolyte, which has proved one of the most nearly successful of any of the attempts to produce sodium by the direct electrolysis of sodium chloride. After trying to use bell-shaped cells made of earthenware for collecting the sodium and separating the anode and cathode products, he attributed the rapid destruction of the cathode bell more to the leakage of the current through the walls of the bell than to the corrosive action of the fused electrolyte. To prevent this, the inventor formed a coating of solid electrolyte over the surface of his "pole cell," which is impervious to the current and which is maintained by cooling the annular metal walls of the cell.

The construction of the apparatus may be readily comprehended from Fig. 3. It consists of a melting pot A closed by the cover D which supports the "pole cell" and the anodes.

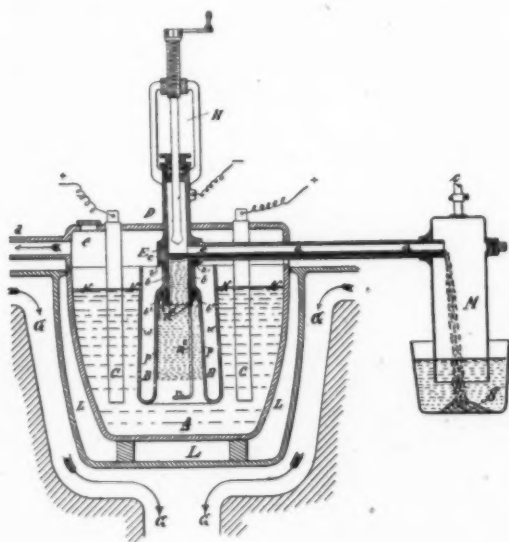


FIG. 3.—GRABAU CELL.

The pole cell consists of the double-walled bell B, the cathode n, an outlet pipe a and the boring arrangement H for removing obstructions in the bell. The electrolyte is solidified on the surface of the bell and the current is forced to go around through the electrolyte instead of through the walls of the bell.

To further facilitate the process, Grabau made use of a mixed electrolyte to reduce the melting point, but this principle is not original with him, for it had already been applied by Bunsen and Matthiessen in 1885. In this way he claims to have obtained an ampere efficiency of 95 per cent. The best results were obtained by the use of a bath of the following composition: $3(\text{NaCl}, \text{KCl}) + \text{SrCl}_2$, that is, one molecule of strontium to three molecules of the alkali chlorides mixed in molecular proportions. The sodium produced contains about 3 per cent of potassium but no traces of strontium. Barium chloride does not melt as easily as strontium chloride, and calcium chloride is so difficult to free from water that he prefers strontium chloride to the other two as the alkaline earth chloride to be used. The chief causes of the commercial failure of the process seem to be the high voltage required and the careful attendance made necessary by the small size of the units and the frequent clogging of the cells.

References:

- Chem. Zeitschr. 1; 409 (1902).
- Chem. Centralblatt (1891, II); 96.
- Zeit. f. angew. Chem. (1890); 336 and (1891); 223.
- U. S. P. No. 464,096 of Dec. 1, 1891.
- U. S. P. No. 464,097 of Dec. 1, 1891.
- U. S. P. No. 465,369 of Dec. 15, 1891.
- Also patented in Ger., Eng., Fr., Italy, Aust., Switz., Belg. and Spain.

Another worker during the year 1886 was Max Sprenger. He suggested the electrolysis of fused salt in a vacuum, but no data are available concerning his results. D. R. P. No. 39,554 of 1886.

Several processes proposed at about this time are not worth special description, but will be briefly mentioned. Bull electrolyzed fused salt and condensed the sodium vapor. Neiwirth vaporized the salt and subjected this vapor to some kind of electrolytic action. Stoerk and Seidler do not seem to have profited by the failure of their predecessors.

References:

- Bull, Eng. P. No. 10,735 of June 7, 1892.
- Seidler, Oesterr.-Ung. Priv. of March 3, 1887.
- Niewerth, Ger. P. No. 65,921 of Jan. 11, 1891; Eng. P. No. 23,773 of 1892.
- Stoerk, Ger. P. No. 68,335 of Aug. 24, 1892; Zeit. f. angew. Chem. (1893); 356.

The next attempt was a retrogression, for the state of the art in 1888 should have enabled Hornung and Kasemeyer to design a more feasible apparatus. No better illustration can be cited which will better show how important it is for an inventor to be familiar with all the work that has previously been done on the subject. Their apparatus consists of a cylindrical containing vessel which acts as the anode and a concentrically arranged iron tube which acted as cathode. The electrolyte was replenished by adding salt through this tube. A circular concentric partition at the top of the tube, dipping a little below the surface of the electrolyte, separated the products of the electrolysis from each other. The required voltage was probably low enough to suit the most exacting commercial limits, but it hardly seems probable that any sodium was ever liberated in this apparatus by electrolysis of fused salt, for the electrodes were in such close proximity that there was nothing to hinder complete depolarization. Even with the easily melted hydroxide it is necessary to have a diaphragm, when the electrodes are near each other, as in the Castner process.

References:

- Ger. P. No. 46,334 of Jan. 29, 1888.
- Zeit. f. angew. Chem. (1889); 216.

An apparatus devised by Rennerfelt in 1893 was a return to the Grabau type of seven years previous. This inventor did not seem willing to concede the point made by Grabau that earthenware is not impervious to the electric current when placed in a bath of fused salt, so he coated the outside of his bell-shaped cathode compartment with a layer of such material. The inside of the bell acted as cathode, instead of having an electrode within the bell, as in the Grabau apparatus. The containing vessel acted as anode and the metal was removed from the bell by suction. The difficulties of this type of apparatus have been shown in the description of the Grabau apparatus. U. S. P. No. 495,600 of April 18, 1893.

In 1893, Borchers designed a furnace on the lines of the best theory of the time, which for a short time held a place in practice, but it could not compete with processes using fused hydroxide. This apparatus was "U" shaped, one leg being considerably larger than the other. The larger leg contained the carbon anode and was made of chamotte to withstand the action of the fused salt and the chlorine. The smaller leg was of iron, and itself formed the cathode; the

two sections were joined together by means of an ingenious arrangement of clamps and a water-cooled ring. The difficulties were the perishability of the apparatus and the high voltage required. The current density at the cathode was 50 amps. per 100 sq. cm. and the fall of potential about 10 volts per cell.

In Becker's "Elektrometallurgie der Alkalimetalle," page 37, the author gives the figures for the cost of plant and the cost of production using the Borchers apparatus. The labor charges are far too low for American conditions, and the amount allowed for depreciation is probably too large, but his figures make the cost per pound of sodium the very high figure of 26 cents. If these figures are accurate, it is very easy to see why this process was not used in practice for any extended period.

References:

Zeit. f. angew. Chem. (1893); 486.

Borchers "Elektrometallurgie," 3d ed., p. 45.

This was followed in 1896 by an apparatus devised by Hanekop, which the inventor claimed to be an improvement on the apparatus of Borchers. He made use of an earthenware melting vessel divided into several parallel compartments by hollow partitions reaching nearly to the bottom. The anodes and cathodes were in separate adjoining compartments. As there is no known fireproof earthenware that will withstand the action of both sodium and chlorine, the apparatus was predestined to failure. The use of a hollow partition between the electrodes was suggested by Grabau, so that both originality and improvements are not in evidence. Ger. P. No. 98,766 of July 4, 1896.

From the work of Rogers, in 1883, no attempt seems to have been made in the United States to electrolyze molten salt directly until 1898. Paul Danckwardt, in this year, attempted to overcome the structural material difficulty by using water jackets for the walls and the partition between the anode and cathode compartments. The bath was kept melted by an oil or gas flame playing upon its surface. This has the disadvantage that a portion of the salt would be converted to carbonate, and the carbonate thus formed would take part in the electrolysis, yielding carbon dioxide at the anode. The red-hot carbon anodes would react with the carbon dioxide and form carbon monoxide, with a correspondingly rapid corrosion of the anode. Another difficulty seems to have been that both the electrodes were located near the bottom of the bath, thus providing the best possible conditions for complete depolarization, as pointed out in the work of Fischer (l. c.). U. S. P. No. 607,506 of July 19, 1898.

Another furnace of American origin appeared in 1902, being the invention of Roepner and Scholl. This furnace was not intended specifically for the electrolysis of sodium chloride, but may be applied, with modifications, to the electrolysis of any fused salt. It consists of a reverberatory furnace on the hearth, of which is a container for the melted bath. One portion of the surface of the bath is exposed to the heat in the furnace, and the remainder is readily accessible from the outside, so that the "removable electrolytic box" can be replaced without interrupting the heating. The chief objects of the invention are to provide external heating in an economical manner and to make all parts replaceable without cooling down the furnace. U. S. P. No. 699,851 of May 13, 1902.

The effort to electrolyze sodium chloride directly seems to have ended in Europe with Borchers in 1893, little or nothing having since been done; but the last attempt in this country was that of Cowles, about 1900. The original feature of this process is the separation of the sodium at a porous carbon cathode, at such a high temperature that the sodium is vaporized. The vapor penetrates through the porous cathode and is condensed in a chamber beneath it, the cathode serving also as the bottom of the fusion chamber, as shown in Fig. 4. U. S. P. No. 679,253 of July 23, 1901.

The commercial separation of metallic sodium by the direct

electrolysis of melted sodium chloride still remains an unsolved problem. It is, however, the most logical manner of reducing the cost of producing sodium, and it is probable that further work will be done along that line, presumably with ultimate success, but the wrecks by the way-side are grim testimonials of the difficulty of the problem. An indirect solution will be discussed under 669,733-374.6, compound processes.

The following references must be added to complete the bibliography of this class:

Beketow, Lumiere Electr. 30; 532 (1888).

Nuernberg Consortorium. Ger. P. No. 160,540 of July 20, 1904.

Matthiesen, Ann. d. Chem. u. Pharm. 93; 277 (1855).

Faraday, Ostwald's "Klassiker." No. 86, pp. 43, 48 and 65.

(To be concluded.)

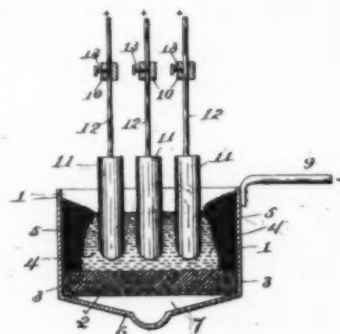


FIG. 4.—COWLES CELL.

Metallurgical Calculations.

By J. W. RICHARDS, PH. D.

Professor of Metallurgy in Lehigh University.

THE BESSEMER PROCESS

The outlines of this famous process are known to every educated person; the mechanical and most of the chemical details are familiar to most technologists; the exact way to run the converter is the source of income to hundreds of superintendents of works, and yet the quantitative side of the chemical and physical operations involved is mastered by very few.

To state the case briefly, melted pig iron is put into the converter, numerous air jets are blown through, the impurities of the iron—carbon, silicon, manganese and, in a special case, phosphorus—oxidize relatively faster than the iron, and the final product is usually nearly pure iron. This is recarburized to steel by spiegel-eisen. During the blow very little free oxygen escapes from the converter, and the gases produced are principally nitrogen, carbon monoxide and some carbon dioxide, while some hydrogen may come from the decomposition of the moisture of the air. The silicon, manganese, phosphorus and iron form silica, manganous oxide (MnO) principally, ferrous oxide (FeO) principally, phosphorus pentoxide, P_2O_5 , which go into the slag, while a little Fe_2O_3 , Mn_2O_3 and SiO_2 escape as fume.

The applications of calculations to this process are numerous and important. They include such subjects as the amount of air theoretically required per ton of iron, the dimensions and power of the blowing engines, the weight of slag produced, the balance sheet of materials, the balance sheet of heat evolved and distributed, the radiation losses, the discussion of the efficiency of the various impurities as heating agents in the process.

AIR REQUIRED.

Basing our calculations on the analysis of the pig iron used, and assuming it to be blown to pure iron, we must next assume the probable loss of iron itself in the blow. This varies considerably, from 1 to 10 per cent on the pig iron used in ordinary practice, but as much as 20 to 25 per cent in some carelessly run "Baby" Bessemer in steel-casting foundries. The silicon all oxidizes to SiO_2 ; iron mostly to FeO , and a small part, say not over one-tenth, to Fe_2O_3 ; manganese mostly

to MnO , a small part, up to one-fourth, may form Mn^2O^3 ; phosphorus forms only P^2O^5 ; carbon burns mostly to CO , but from one-fifth to, at times, nearly one-half, burns to CO^2 . When all the calculated oxygen has been found, the blast to contain it can be calculated, if it is assumed that no free oxygen escapes from the converter; at times, however, up to one-third of all the oxygen blown in may thus escape, but this is very exceptional, ordinarily less than one-fifth thus escapes, and often none at all.

Problem 61.

Pig iron containing 3.10 per cent carbon, 0.98 silicon, 0.40 manganese, 0.101 phosphorus and 0.06 sulphur is blown in an acid-lined converter, to metal practically free from carbon, silicon and manganese, but no sulphur or phosphorus is eliminated. To get the minimum and the maximum amounts of air which could be needed, make the following assumptions:

	Case 1.	Case 2.
Per cent of iron lost by oxidation.....	1	15
Proportion of iron oxidized to Fe^2O^3	none	one-tenth
Proportion of Mn oxidized to Mn^2O^3	none	one-fifth
Proportion of C oxidized to CO^2	one-fifth	one-half
Proportion of O^2 escaping in the gases.....	none	one-third

Requirement: (1) Find the weight of dry air needed per metric ton of pig iron blown, in each case, and its volume at 0°C . Express the results also in pounds and cubic feet per ton of 2,000 pounds.

Solution:

Case 1.

Oxygen needed per 1,000 kg. of pig iron:

C to CO^2	6.2 kg.	$\times \frac{32}{12} =$	16.53 kg.
C to CO	24.8 "	$\times \frac{16}{12} =$	33.07 "
Si to SiO^2	9.8 "	$\times \frac{32}{28} =$	11.20 "
Mn to MnO	4.0 "	$\times \frac{16}{55} =$	1.16 "
Fe to FeO	10.0 "	$\times \frac{16}{56} =$	2.86 "
			64.82 "
N^2 accompanying =			216.07 "

Air needed	= 280.89 "
Volume at 0°C	= 217.2 m^3
Volume needed per 1000 oz. Av.....	= 217.2 ft^3
Volume needed per 2000 lbs. Av.....	= 6,950 ft^3

Case 2.

Oxygen needed per 1000 kg. of pig iron:

C to CO^2	15.5 kg.	$\times \frac{32}{12} =$	41.33 kg.
C to CO	15.5 "	$\times \frac{16}{12} =$	20.67 "
Si to SiO^2	9.8 "	$\times \frac{32}{28} =$	11.20 "
Mn to MnO	3.2 "	$\times \frac{16}{55} =$	0.93 "

Mn to Mn^2O^3	0.8 "	$\times \frac{48}{110} =$	0.34 kg.
Fe to FeO	135.0 "	$\times \frac{16}{56} =$	38.57 "
Fe to Fe^2O^3	15.0 "	$\times \frac{48}{112} =$	6.43 "
O^2 unused (one-half sum of above).....			59.73 "
			179.20 "
N^2 accompanying =			597.33 "
Air used			776.53 "
Volume at 0°C			600 m^3
Volume used per 1000 oz. Av.....			600 ft^3
Volume used per 2000 lbs. Av.....			19,200 ft^3

For temperatures of the air above 0°C , a corresponding increase in the volume used would be found. Since this is net air received by the converter an allowance for loss of 10 to 25 per cent (in exceptional cases 50 per cent) would be needed to get the piston displacement of the blowing engines. The above figures are the maximum and minimum for this quality of pig iron only, blown in an acid-lined converter; other qualities of pig iron might require a little more or less, and if blown in a basic-lined converter considerably more, to oxidize the phosphorous. The detailed calculations can be made in each specific case.

AIR RECEIVED.

The converse of the preceding proposition is to take an actual case, in a Bessemer converter, and to calculate how much air is being received. This will serve as a check on the blowing engines, since the volume received, divided by the piston displacement, gives the volume efficiency of the blowing plant. To make the calculation we need to know the weight and analysis of the pig iron and the analysis of the blown metal, in order to find the weights of impurities oxidized, also the average composition of the escaping gases, to find the proportion of carbon burning to CO^2 and of unused oxygen; also the composition of the slag, to get therefrom the weight of iron oxidized and the weight of slag, if practicable, but this can sometimes be calculated; also the weight and composition of the fume, if it is considerable. The temperature of the air entering the blowing cylinders, its hygrometric condition and the barometric pressure, should also be noted.

Problem 62.

At the South Chicago works of the Illinois Steel Co. (see paper by Howe, in Trans. American Institute of Mining Engineers, XIX. [1890-91], p. 1,127), the charge weighed 22,500 pounds, and contained 2.98 per cent carbon, 0.94 silicon, 0.43 manganese, 0.10 phosphorus, and 0.06 sulphur. After blowing 9 minutes 10 seconds the bath contained 0.04 per cent carbon, 0.02 silicon, 0.01 manganese, 0.11 phosphorus and 0.06 sulphur. The slag formed contained 63.56 per cent silica, 3.01 alumina, 21.39 FeO , 2.63 Fe^2O^3 , 8.88 MnO , 0.90 CaO and 0.36 MgO , of which the Al^2O^3 , CaO and MgO and part of the SiO^2 come from the lining. The gases, analyzed dry, show an average composition during the blow of

CO^2	5.20 per cent.
CO	19.91 "
H^2	1.39 "
N^2	73.50 "

and were free from fume.

The piston displacement during the blow was 190,406 cubic feet, air in engine room 36°C , humidity 50 per cent, barometer 756 millimeters.

Requirements: (1) The weight of oxygen acting on the bath during the blow, and the theoretical volume of air at standard conditions to which this would correspond, per 2,000 pounds of metal blown.

(2) The volume of moist air at the conditions of the engine room, received by the converter during the blow, and the volume efficiency of the blowing machine and connections.

(3) The weight of slag produced and the loss in weight of the lining by corrosion during the blow.

Solution: (1) The percentages of impurities left in the bath are so small that we can take them as equivalent to the same percentages reckoned on the original weight of the bath. If they had been larger it would be necessary to assume an approximate loss of iron during the blow, find the final weight of the bath and reckon the percentages on this revised weight.

Making this assumption, we know at once the weights of carbon, silicon and manganese oxidized, but we do not know the weight of iron lost. That follows, however, from a consideration of the slag, for the manganese and iron in the slag are derived only from the metallic bath, and the analysis of the slag practically gives us the relation between the weights of manganese and iron in it; since we know the weight of the former oxidized, the weight of iron lost can be calculated. Thus the slag contains:

MnO	8.88 per cent	=	6.88 per cent	Mn
FeO	21.39 "	=	16.64 "	Fe as FeO
Fe ² O ³	2.63 "	=	1.84 "	Fe as Fe ² O ³

The loss of manganese being 0.42 per cent = 94.5 pounds, the loss of iron is:

$$94.5 \times \frac{16.64}{6.88} = 228.6 \text{ pounds Fe as FeO}$$

$$94.5 \times \frac{1.84}{6.88} = 25.3 \text{ " Fe as Fe}^2\text{O}^3$$

The remaining item still undetermined is the weight of carbon oxidizing to CO² and to CO. The gas analysis shows 5.20 per cent CO² to 19.91 per cent CO², and since equal volumes of each of these gases contain equal weights of carbon,

it follows that $\frac{5.20}{25.11}$ of the total carbon is present in the

gas as CO², and the rest as CO. Since the total carbon oxidized is $22,500 \times 0.0294 = 661.5$ pounds, we have

$$661.5 \times \frac{5.20}{25.11} = 137.0 \text{ pounds C burning to CO}^2$$

$$= 524.5 \text{ " " CO}$$

Weight of oxygen absorbed by the bath:

C to CO ²	137.0 pounds	$\times \frac{8}{3}$	=	365.3 pounds
C to CO.....	524.5 "	$\times \frac{4}{3}$	=	699.3 "
Si to SiO ²	207.0 "	$\times \frac{32}{28}$	=	236.6 "
Mn to MnO....	94.5 "	$\times \frac{16}{55}$	=	27.5 "
Fe to FeO....	228.6 "	$\times \frac{16}{56}$	=	65.3 "
Fe to Fe ² O ³ ...	25.3 "	$\times \frac{48}{112}$	=	10.8 "

$$\text{N}^2 \text{ corresponding} \dots \dots \dots = 4682.7 \text{ "}$$

$$\text{Dry air corresponding} \dots \dots \dots = 6087.5 \text{ "}$$

$$\text{Volume at 0}^\circ \text{ C.} \dots \dots \dots = 75.483 \text{ ft}^3$$

$$\text{Weight of O}^2 \text{ per 2000 pounds} \dots \dots \dots = 124.9 \text{ lbs. (1)}$$

$$\text{Volume of air per 2000 pounds} \dots \dots \dots = 6,710 \text{ ft}^3 \text{ (1)}$$

(This result is for comparison with data of Problem 61).

(2) The nitrogen in the gases can be obtained from its volume relation to the carbon, and from this we can calculate the real volume of blast used.

Weight of carbon in 1 cubic foot of gases:

$$(0.0520 + 0.1991) \times 0.54 = 0.1356 \text{ oz. Av.}$$

Volume of gases produced at standard conditions:

$$\frac{661.5 \times 16}{0.1356} = 78,057 \text{ ft}^3$$

Volume of nitrogen at standard conditions:

$$\frac{78,057 \times 0.7350}{57,372 \times 1.26} = \frac{57,372 \text{ ft}^3}{72,289 \text{ oz. Av.}} = 4,518 \text{ lbs.}$$

The question now is, how much nitrogen is contained in each cubic foot of air in the engine room. Knowing that, we are prepared to calculate the volume of this actually received by the converter:

$$\begin{array}{ll} \text{Barometric pressure} & = 756 \text{ m. m.} \\ \text{Tension of moisture } (44 \times 0.5) & = 22 \text{ "} \end{array}$$

$$\text{Tension of air present} = 734 \text{ "}$$

$$\text{Tension of nitrogen present } (734 \times 0.792) = 580 \text{ "}$$

Weight of nitrogen in 1 cubic foot:

$$1.26 \times \frac{273}{273 + 36} \times \frac{580}{760} = 0.8495 \text{ oz. Av.}$$

Volume of air actually received:

$$\frac{4,518 \times 16}{0.8495} = 85,095 \text{ ft}^3 \text{ (2)}$$

Volume efficiency of machinery:

$$\frac{85,095}{190,406} = 0.447 = 44.7 \text{ p. c. (2)}$$

(3) The slag contains 8.88 per cent of MnO, equal to 6.88 per cent of Mn, as already calculated. But 94.5 pounds of manganese is oxidized, therefore, the weight of slag produced is:

$$94.5 \div 0.0688 = 1374 \text{ pounds. (3)}$$

$$\text{Weight of SiO}^2 \text{ in slag } (1374 \times 0.6356) = 873 \text{ pounds}$$

$$\text{SiO}^2 \text{ from the Si of bath } (207.0 + 236.6) = 444 \text{ "}$$

$$\text{SiO}^2 \text{ corroded from the lining} = 429 \text{ "}$$

$$\text{CaO, Al}^2\text{O}^3 \text{ and MgO } (1374 \times 0.0427) = 59 \text{ "}$$

$$\text{Loss in weight of lining} = 488 \text{ " (3)}$$

BLAST PRESSURE.

It is necessary to use sufficient blast pressure to overcome the static pressure of the metallic bath, plus that of the slag formed, also the back pressure in the converter, to give the necessary velocity to the air in the tuyeres and to overcome friction in the same. When the tuyeres are near to the surface of the bath, pressures of 1 or 2 pounds will run the small converter, but the ordinary converter with bottom tuyeres requires from 15 to 30 pounds pressure per square inch (1.054 to 2.108 kg. per square c. m.). We will consider the latter, the more frequent and the more complex case to discuss.

The metal lies 12 to 24 inches (30 to 60 c. m.) deep in the converter. Since its specific gravity melted is about 6.88 (Roberts and Wrightson), the ferro-static pressure which it exerts is practically 0.25 pounds per square inch for each inch depth of metal, or 0.00688 kilos. per square centimeter for each centimeter depth.

The slag lying on the metal has a specific gravity melted of approximately half that of the metal. Its amount may vary from 5 to 10 per cent of the weight of the metal treated, in an acid-lined converter, up to from 15 to 35 per cent in a basic-lined vessel. Taking into account its lower specific gravity, its depth in the converter may be, therefore 10 to 20 per cent the depth of metal in an acid-lined vessel, and 30 to 70 per cent in a basic-lined converter; but the static pressure exerted would be only in direct proportion to the relative weights; i. e., 5 to 10 or 15 to 35 per cent of that exerted by the metal.

The static pressure of the slag may, therefore, be reckoned as 0.125 pounds per square inch for each inch in depth, or 0.00344 kilos. per square centimeter for each centimeter depth, and the depth of slag as lying between the following extremes.

		Acid Lined.	Basic Lined.
Depth of metal	Inches	12 to 24	12 to 24
	Centimeters ...	30 to 60	30 to 60
Depth of slag	Inches	1.2 to 4.8	3.6 to 16.8
	Centimeters ..	3.0 to 12.0	9.0 to 42.0

The probable depth of slag can be calculated in any particular case, when the composition of the metal to be blown is known, its approximate depth in the vessel, and the approximate composition of the slag to be formed.

The back pressure of gases in the converter itself, that is, their static pressure, will vary with the shape of the converter and the size of the free opening for their escape into the air. A measurement at the Pennsylvania Steel Co's works gave 0.275 pounds per square inch, but it is not stated just how the measurement was made. If we know approximately the volume of gas which must escape from the converter and from its temperature and the time and the size of the outlet calculate its velocity, the static pressure giving it this velocity can be calculated as

$$h = \frac{V^2}{2g}$$

in which, if V is in feet per second, $2g = 64.3$ and the resultant pressure is in feet of the hot gas; if V is in meters per second, $2g = 19.6$, and h is in meters of the hot gas. Knowing the approximate specific gravity of the hot gas (weight of 1 cubic foot in pounds or of 1 cubic meter in kilograms) the static pressure is obtainable in pounds per square foot or kilograms per square meter.

Illustration: The gases escaping from a converter are 78,057 cubic feet (standard conditions), and weigh 0.0801 pounds per cubic foot (standard conditions). They escape from the converter at an average temperature of 1,500° C., and the opening is 24 inches in diameter. What is the gaseous back pressure in the converter? Time of blow 9 minutes 10 seconds.

Volume of gas at 1,500°:

$$78,057 \times \frac{1500 + 273}{273} = 506,940 \text{ ft}^3$$

Volume per second:

$$506,940 \div 550 = 921.7 \text{ "}$$

Area of outlet:

$$2 \times 2 \times 0.7854 = 3.1416 \text{ ft}^2$$

Velocity (assuming 0.9 coefficient):

$$921.7 \div 0.9 \div 3.1416 = 326 \text{ ft. per second}$$

Head of hot gas giving velocity:

$$h = \frac{326 \times 326}{64.3} = 1,653 \text{ ft.}$$

Pressure of this column per square foot:

$$1.653 \times \frac{273}{1773} \times 0.0801 = 20.4 \text{ pounds}$$

$$\text{Per square inch} = 0.14 \text{ "}$$

This solution omits a consideration; the velocity of the gases in the body of the converter is neglected. This is somewhat counterbalanced by the great friction of the gases against the sides of the converter, so that the one item tends to neutralize the other. If the interior were 8 feet in diameter, the velocity of the gases therein would average only some 20 feet per second, showing the above corrections to be practically negligible, since the pressure thus represented would be only 0.4 per cent of the total obtained above.

The pressure necessary to force the blast through the tuyeres is calculable on principles similar to the above; the

differences are that the blast, at temperatures varying from 100° C. in the blast box to possibly 200° at its entrance into the metal, is divided up into fifty or 150 streams of approximately 1 centimeter (0.4 inch) in diameter, the length of tuyere being some 50 centimeters (20 inches). The formula similar to that used for chimney draft, or rather, frictional resistance in a chimney, applies to this case.

$$h = \frac{V^2}{2g} \frac{KL}{D}$$

in which h is the head in terms of the air passing, V is its velocity, $2g$ the gravitation constant, L the length of the tuyere, D its diameter, and K the coefficient of friction, which latter is for relatively smooth flues 0.05 (Grashof), and may be so assumed here.

Problem 63.

In the converter mentioned in Problem 62, where 22,500 pounds of metal was blown in 9 minutes 10 seconds, using as therein calculated 85,095 cubic feet of air, at 36° C., and producing 1,374 pounds of slag, assume the inside diameter of the converter as 7 feet, and that the bottom contains fourteen tuyere blocks, each containing eleven openings of 0.5 inch diameter each; blocks 24 inches long. Assume back pressure in converter 0.14 pounds per square inch, total blast pressure in equalizing reservoir 27 pounds per square inch. Temperature of air in the tuyeres 150° C.

Required: (1) The pressure needed to overcome the head of metal and slag.

(2) The pressure absorbed in friction in the tuyeres.

(3) The pressure represented by the velocity of the blast in the tuyeres.

(4) The loss of pressure from the reservoir to the blast-box.

(5) The distribution of the total pressure.

(6) The length of the blow if the blast pressure were reduced to 20 pounds.

(7) The length of the blow if the pressure were maintained at 27 pounds, but twenty-one tuyere blocks (each with eleven 1/2-inch holes) were used.

Solution: (1) At the start there is 22,500 pounds of melted metal, the volume of which will be

$$\frac{22,500}{6.88 \times 62.5} = \frac{22,500}{430} = 52.3 \text{ cubic feet}$$

The depth of metal, the inside diameter being 7 feet, is

$$\frac{52.3}{7 \times 7 \times 0.7854} = \frac{52.3}{38.5} = 1.356 \text{ feet}$$

$$= 16.4 \text{ inches}$$

$$\text{Static pressure} = 16.4 \times 0.25 = 4.1 \text{ lbs. per sq. in.}$$

The slag, formed during the first half of the blow, weighs 1,374 pounds, and has a volume of

$$\frac{1374}{3.44 \times 62.5} = \frac{1374}{215} = 6.4 \text{ cubic feet}$$

The depth of slag, at its maximum, will be

$$\frac{6.4}{38.5} = 0.167 \text{ feet}$$

$$= 2.0 \text{ inches}$$

$$\text{Static pressure } 2.0 \times 0.125 = 0.25 \text{ lbs. per sq. in.}$$

The static pressure during the blow will, therefore, be 4.1 pounds to start with, increasing during the first half of the blow to 4.35 pounds, and staying practically constant at that, and, therefore, will average

$$\frac{4.1 + 4.35}{2 \times 2} = \frac{4.35}{2} = 4.29 \text{ pounds} \quad (1)$$

(2) Each of the $14 \times 11 = 154$ tuyeres receives $85,095 \div 154 \div 550 = 1.005$ cubic feet of air per second, measured at 36° C. At 150° C. this volume is

$$1.005 \times \frac{273 + 150}{273 + 36} = 1.375 \text{ cubic feet}$$

And the velocity in the tuyere:

$$1.375 \div \frac{0.5 \times 0.5 \times 0.7854}{144} = 1009 \text{ feet per second}$$

The head absorbed in friction in the 24-inch tuyeres will be

$$h = \frac{1009 \times 1009}{64.3} \times \frac{0.05 \times 2}{0.0417} = 37.893 \text{ feet}$$

Changing this pressure of air at 150° C. to pounds per square inch we have:

$$\begin{aligned} \text{Weight of 1 cubic foot of air at } 0^\circ &= 0.0808 \text{ pounds} \\ \text{Weight of 1 cubic foot of air at } 150^\circ &= 0.0522 \text{ " } \\ \text{Weight of air column} &= 37.893 \times 0.0522 = 1.978 \text{ " } \\ \text{Pressure in pounds per square inch} &= 13.70 \text{ " (2)} \end{aligned}$$

(3) The pressure absorbed as velocity has already been expressed in getting the friction in the tuyeres. The velocity head is simply:

$$h = \frac{V^2}{2g} = \frac{1009 \times 1009}{64.3} = 15.835 \text{ feet}$$

which becomes in pressure

$$\begin{aligned} 15.835 \times 0.0522 &= 826.6 \text{ pounds per square foot} \\ &= 5.73 \text{ pounds per square inch (3)} \end{aligned}$$

(4) The remaining part of the 27 pounds pressure used is lost between the blast reservoir and the entrance to the tuyeres. It is

$$27.00 - (13.70 + 5.73 + 4.29 + 0.14) = 3.14 \text{ pounds (4)}$$

(5) Distribution of blast pressure:

Fall between reservoir and tuyeres	= 3.14 pounds	= 11.6%
Absorbed in friction in the tuyeres	= 13.70 " "	= 50.7%
Absorbed in velocity in the tuyeres	= 5.73 " "	= 21.2%
Static head of liquid bath	= 4.29 " "	= 15.9%
Velocity head of issuing gases	= 0.14 " "	= 0.6%
	27.00 " "	= 100.0%

(6) All the items of absorption of pressure are proportional to the square of the velocity of the gases, excepting the static pressure of the bath. It remains constant at 4.29 pounds. If the total pressure were reduced to 20 pounds, there would be only 20 - 4.29 = 15.71 pounds pressure to give velocity and overcome friction, instead of 27 - 4.29 = 22.71 pounds. The relative quantities of air blown through in a given time in the two instances would be practically proportional to the square roots of the two effective pressures, *i. e.*:

$$\sqrt{22.71} : \sqrt{15.71} = 1 : 0.832$$

And the times of the blows inversely as the latter:

$$\begin{aligned} 550 \text{ sec.} \div 0.832 &= 673 \text{ seconds} \\ &= 11 \text{ min. 13 sec. (6)} \end{aligned}$$

(7) If the tuyere area were increased 50 per cent, then the velocity of the air in the tuyeres would be decreased one-third, assuming the amount of air passing to be unchanged. This would decrease the pressure absorbed in friction, and in giving velocity in the tuyeres to $(0.67)^2 = 0.444$ of its former amount. The 19.43 pounds previously absorbed in these two items would then become $19.43 \times 0.444 = 8.61$ pounds, and the total pressure needed to run the converter just as fast as before would be 27 - (19.43 - 8.61) = 16.18 pounds per square inch. If, however, the pressure were maintained at 27 pounds, giving still 27 - 4.29 = 22.71 pounds to overcome frictional resistances and to give velocity, then the velocity and consequent amount of air blown through by this 22.71 pounds pressure would increase in proportion to the square roots of these two available pressures; *i. e.*, be as

$$\sqrt{16.18} - 4.29 : \sqrt{22.71} - 4.29 = 1 : 1.38$$

The duration of the blow would be just that much shorter; *i. e.*:

$$\begin{aligned} 550 \text{ sec.} \div 1.38 &= 398 \text{ seconds} \\ &= 6 \text{ min. 38 sec. (7)} \end{aligned}$$

FLUX AND SLAG.

No flux is used in the acid-lined converter, and the silica, iron oxides and manganese oxide formed in the converter unite to a silicate slag which corrodes the lining and thus takes up more silica. The slag being analyzed, its weight is obtained by considering the percentage of manganese which it contains, because the weight of manganese oxidized is known definitely from the analysis of the bath; it is usually all oxidized. Calculation of the weight of slag cannot be based upon the silica, because an unknown amount comes from the lining; nor upon the iron, because the weight of iron left in the converter is not definitely known. Having the weight of the slag, analysis tells us the total weight of silica in it, as also the amount of iron. The silica in the slag minus that formed from silicon in the pig iron, gives silica corroded from the lining.

In the basic Bessemer converter, phosphorus is nearly entirely eliminated from the metal, so that, assuming none to be volatilized, the amount going into the slag is known, and using the slag analysis the weight of slag can be calculated. In this process the lining is mainly dolomite, containing CaO and MgO, in proportions easily determined by analysis. The weight of slag being known, the amount of corrosion of the lining can be determined from the percentage of magnesia therein, which may be assumed as practically coming entirely from the lining; it cannot be told from the CaO in the slag, because nearly pure CaO is added during the blow, and some of it, a variable amount, gets blown out of the converter. For the same reason it is not possible to base a good calculation of the weight of slag on the lime alone which is added, because of the indefinite proportion of it which is blown out. The weight of slag may also be gotten from the silica or manganese oxide in it, assuming these to come almost entirely from the oxidation of silicon or manganese.

Lime must be added as flux, in the basic converter, to protect the lining and to make the slag so basic that the percentage of silica in it is below 15 per cent, phosphoric acid below 20 per cent, and lime over 50 per cent. These considerations must be balanced in each particular case.

Illustration: Pig iron blown in a basic-lined converter contained 1.22 per cent silicon, 2.18 phosphorus, 1.03 manganese and 3.21 carbon. It is blown until all of these and 2.00 per cent of iron are oxidized, and burnt lime is added to form slag during the blow. Composition of the burnt lime: MgO, 1.00 per cent; SiO₂, 2.00 per cent; CaO, 97 per cent. How much lime should be added per 10 metric tons of pig iron charged?

The slag-forming ingredients from the oxidation of the bath, and the addition of X kilos. of lime, are

SiO ₂	10,000 × 0.0122 × 60/28	= 261.4 kg.
P ₂ O ₅	10,000 × 0.0218 × 142/62	= 499.3 "
MnO	10,000 × 0.0103 × 71/55	= 133.0 "
FeO	10,000 × 0.0200 × 72/56	= 257.1 "
CaO	X × 0.9700	= 0.97 X
MgO	X × 0.0100	= 0.01 X
SiO ₂	X × 0.0200	= 0.02 X

$$\text{Weight for slag} = X + 1150.8 \text{ kg.}$$

Corrosion of the lining will undoubtedly increase this weight, so some allowance should be made, say to increase it 5 per cent, probably an outside figure. Of this 5 per cent, half can be considered lime and half magnesia. The total weight of slag will then be 1.05 X + 1208.3, and of the ingredients principally in question:

$$\begin{aligned} \text{SiO}_2 &= 261.4 \text{ kg.} + 0.02 X \\ \text{MgO} &= 28.7 \text{ " } + 0.035 X \\ \text{CaO} &= 28.7 \text{ " } + 0.995 X \end{aligned}$$

To make our slag 50 per cent CaO will require the addition of enough to make

$$28.7 + 0.995 X = 0.50 (1.05 X + 1208.3) \\ X = 1224 \text{ kg.}$$

To make a slag with at most 15 per cent of SiO_2 requires

$$261.4 + 0.02 X = 0.15 (1.05 X + 1208.3) \\ X = 583 \text{ kg.}$$

To make a slag with at most 20 per cent of P_2O_5 requires

$$499.3 = 0.20 (1.05 X + 1208.3) \\ X = 1227 \text{ kg.}$$

The larger of these three amounts would be used, with 10 per cent added to cover lime dust blown out, making 1350 kg. added, and the calculated composition of the slag:

CaO	1250 kg.	= 50.1 per cent
MgO	723 "	= 2.9 "
SiO_2	286 "	= 11.4 "
P_2O_5	499 "	= 20.0 "
FeO	257 "	= 10.3 "
MnO	133 "	= 5.3 "

Total 2497 "

RECARBURIZATION.

When the bath has been blown to nearly pure iron, melted spiegeleisen is run in, to add the necessary carbon and manganese. Knowing the approximate composition and weight of the bath, and the composition of the melted spiegel, a simple arithmetical calculation would give the amount of the latter to be added, assuming no loss of carbon or manganese in the operation. But experience shows that there is some loss, and that the carbon and manganese in the finished metal are always lower than the calculated amount. An interesting field is open here for calculating the loss of manganese and carbon and the amount of oxygen which must have been in the metal to cause these losses. A tabulation of many such calculations gives the metallurgist the necessary data for assuming the average amounts of carbon and manganese lost during recarburization, under different conditions of working, such as letting the metal stand before pouring or pouring at once, turning on the blast 5 or 10 seconds to mix up the bath, etc.

Problem 64.

At the end of the blowing the converter of Problem 62 contained 21,283 pounds of metal of the composition 0.04 per cent carbon, 0.02 silicon, 0.01 manganese, 0.11 phosphorus, 0.06 sulphur, an unknown amount of oxygen (probably < 0.3 per cent) and the rest iron. There is added to it 2,500 pounds of spiegeleisen, containing 4.64 per cent carbon, 0.035 silicon, 14.90 manganese, and 0.139 phosphorus. The finished metal contained 0.45 per cent of carbon, 0.038 silicon, 1.15 manganese, 0.109 phosphorus and 0.059 sulphur. Assume no iron oxidized.

Required: (1) A balance sheet of materials before and after recarburizing.

(2) The proportions of carbon and manganese going into the finished metal.

Solution: (1)

	Blown Metal.	Spiegel.	Steel.	Gases or Slag.
C	8.5	116.0	106.5	18.0
Si	4.3	0.9	9.0	— 3.8
Mn	2.1	372.5	271.2	103.4
P	23.4	3.5	25.8	1.1
S	12.8	14.0	— 1.2
Fe	21,232	2,007	23,239

The differences in the sulphur, phosphorus and silicon are within the limits of error of the data, but there is no doubt as to the loss of carbon and manganese.

(2) The proportions of the two elements in question going into the finished steel are:

$$\text{Carbon} \dots\dots\dots 106.5 \div 124.5 = 0.85 = 85 \text{ per cent} \\ \text{Manganese} \dots\dots\dots 271.2 \div 374.6 = 0.72 = 72 \text{ "}$$

The calculated percentages in the finished steel should have been, and actually were:

$$\text{Carbon} \dots\dots\dots 0.53 - 0.45, \text{ loss} = 0.08 \text{ per cent} \\ \text{Manganese} \dots\dots\dots 1.58 - 1.15, \text{ loss} = 0.43 \text{ "}$$

Concerning oxygen removed, if we assume the loss of carbon and manganese to be due to their combining with oxygen dissolved in the bath, to form CO and MnO, the percentage of oxygen thus absorbed is:

$$\text{By carbon} \dots\dots\dots 0.08 \times 16/12 = 0.11 \text{ per cent} \\ \text{By manganese} \dots\dots\dots 0.43 \times 16/55 = 0.12 \text{ "}$$

0.23 "

[The next instalment of these calculations will consider the thermo-chemistry of the Bessemer process.]

The Influence of Nickel and Carbon on Iron.*

By G. B. WATERHOUSE.

The object of the following research is to help in the study of the ternary alloys of iron, carbon, and another metal or metalloid, by considering a series of steels of constant nickel with varying carbon percentages. The other elements, which are present of necessity, were kept as low and constant as possible.

The materials employed by the author in preparing the steels were puddled bar-iron, known as muck bar; wood charcoal, and an especially pure sample of electrolytically deposited nickel. By analysis it was found to contain 99.47 per cent nickel. The nickel was melted in a crucible, thoroughly mixed, and shotted to get it in a form suitable for weighing. It was decided to make a series of nine ingots, with the nickel constant at about 3.5 per cent and the carbon rising from 0.40 per cent, which was the lowest possible, using graphite crucibles, to about 1.60 per cent.

The muck bar, as may be seen in the chemical section, compares very favorably with the best Swedish iron; 89.5 pounds of it were charged into the crucible together with the calculated amount of charcoal. The furnace was a Siemens regenerative one, gas fired. When clear melted 3.5 pounds of the shotted nickel were added. Before teeming 0.5 of an ounce of aluminium was thrown in, the steel poured into a second hot crucible to ensure mixing, and then into a $\frac{1}{4}$ -inch square mould. When topped the ingots showed no sign of blow-holes or piping, and the surface was good and smooth.

The fractures were so different from those of ordinary steel that they quite deceived the inspector, who usually grades the steel very accurately. After topping, the ingots were reheated, hammered down to $1\frac{1}{2}$ square inch billets, and finally reheated and rolled to 1-inch diameter round bars.

An extremely interesting point was the entire absence of any difficulty due to a thick and clinging scale. This scale is mentioned by Mr. E. Windsor Richards in the discussion following the reading of Mr. Wiggin's paper. He considered it to be one of the great difficulties attending the use of nickel steel, which was then a new material, and in the case of plates it produced a rough surface. Mr. F. W. Paul, speaking in the same discussion, had experienced the same difficulty, especially with plates over $\frac{1}{4}$ inch thick. Mr. J. G. Eaton, of the United States Navy, had also noticed the same phenomenon, the rolling in of the scale producing a series of ridges on the under surface. The difficulty has not been met with much in

* A portion of this article is reproduced from a paper by the author, presented to the Iron and Steel Institute in September, 1906. The above article contains, however, considerable new matter.—Editor.

America, owing, probably, to the high manganese contents of the steels. As will be shown later, the series of steels examined in this research are very low in manganese. It is interesting to find that a small amount of aluminium has taken the place of the manganese, and that without any special precautions the steels rolled perfectly.

The plan of research was to take the bars prepared in the above manner and to examine them thoroughly, first obtaining their complete chemical analyses, and then testing them mechanically under several conditions of heat treatment. Micrographic analyses under these conditions were to be also made, and the heating and cooling curves determined. The results may be conveniently classified under the heads chemical, mechanical, microscopical and pyrometric.

CHEMICAL.

Before the ingots were worked drillings were taken from them at a depth of 1 inch near to the top. The carbon was estimated, and the results which were obtained at the works at Syracuse are given in the first column of Table I. The other results are those found by the author in the metallurgical laboratory, Columbia University, on drillings taken from three to five places in the rolled bars, and well mixed. They are arranged in order of ascending carbon. Through an error at the mill, O and P were mislettered, so that they do not come in alphabetical order.

It is gratifying to notice the low sulphur. In the case of a coke-fired furnace it would not have been lower than 0.025 to 0.030 per cent, even with the best coke, because the steels were in the furnace for an average of 4½ hours.

The carbon in the upper members of the series existed in

ite; it is insoluble in boiling hydrochloric or nitric acid. The microscope shows it to occur in an amorphous condition. The results of Table III. show that it exhibits a tendency to recombine with the iron when heated under oxidizing conditions, so that it meets Prof. Ledebur's three requirements for temper carbon.

Two pieces of "T" were taken in the rolled condition, and after submitting to treatment "A," deeply notched at intervals of 1 inch, and the Metcalf experiment performed. There were seven segments, the first was brought to a welding heat, and the last was at a low hardening heat. After quenching in cold, salt water, they were fractured and the temper graphite determined in the various segments. There was a trace in the first segment of the rolled bar, and a stronger trace in the bar previously annealed. This increased progressively, but at the sixth segment amounted to only .23 per cent and .30 per cent, respectively. The fracture at the last segment was just off color in the rolled bar, and decidedly dark in the other sample. The experiment shows that the temper graphite readily passes into solution by heating above Ar. 1.

MECHANICAL.

Definitions: Treatment A.—Pieces of each bar, 12 inches long, were placed in a gas-fired muffle, and raised to 1,000° C. They occupied 1 hour 25 minutes in reaching this temperature. For 25 minutes they were maintained at this heat, then removed and cooled in air. The object of this procedure was to recrystallize the steels, and to remove any distortion or strain produced during the rolling. The treatment corresponds to the "normalizing" used by Prof. Arnold in his researches, so that the results given in this paper are comparable with his.

TABLE I

Mark of Steel.	Muck Bar.	K.	L.	M.	N.	P.	O.	Q.	R.	S.	T.
Carbon in ingot.....	0.40	0.49	0.61	0.82	0.93	1.00	1.23	1.46	1.62	1.83
Total carbon in bar.....	0.12	0.41	0.51	0.63	0.79	0.92	0.97	1.24	1.54	1.64	1.82
Nickel.....	3.79	3.79	3.76	3.81	3.79	3.75	3.81	3.82	3.82	3.79
Silicon.....	0.055	0.102	0.103	0.133	0.129	0.126	0.095	0.117	0.144	0.151	0.133
Manganese.....	trace	0.086	0.054	0.05	0.051	0.047	0.058	0.057	0.054	0.047	0.051
Sulphur.....	0.008	0.013	0.014	0.014	0.014	0.014	0.014	0.015	0.015	0.014	0.016
Phosphorus.....	0.007	0.007	0.008	0.008	0.008	0.008	0.008	0.008	0.007	0.009	0.009
Aluminium.....	0.01	0.01	0.009	0.01	0.009	0.011	0.009	0.01	0.01	0.01

two forms, combined and graphitic. The percentages are given in Table II.

For the definitions of the treatment the steel has undergone reference may be made to the mechanical section.

TABLE II.

No.	Rolled.		Submitted to Treatment A.		Submitted to Treatment B.	
	Combined Carbon.	Graphitic Carbon.	Combined Carbon.	Graphitic Carbon.	Combined Carbon.	Graphitic Carbon.
Q	1.24	1.24	1.24
R	1.48	0.04	1.21	0.31	0.50	1.02
S	1.13	0.51	0.93	0.71	0.21	1.43
T	1.02	0.80	0.91	0.91

The following results were obtained by Dr. Mathews:

TABLE III.

No.	Billets.		Rolled.		Submitted to Treatment B.	
	Combined Carbon.	Graphitic Carbon.	Combined Carbon.	Graphitic Carbon.	Combined Carbon.	Graphitic Carbon.
Q	1.24	1.24	1.24
R	1.01	.47	1.44	.04	.43	1.05
S	1.09	.57	1.15	.51	.16	1.50
T	1.00	.83	1.03	.80	.10	1.73

The graphitic carbon of these steels corresponds to the temper carbon of Prof. Ledebur, now known as temper graph-

Treatment B.—The bars were packed in a wrought-iron tube, placed in an annealing furnace at 500° C., and brought to a temperature of 870° C. to 880° C. in 5½ hours. They were kept at this temperature for 3 hours, then the tube was with-

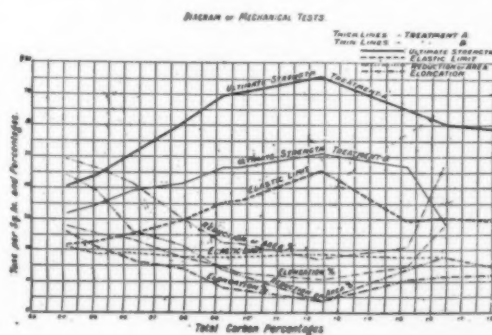


FIG. 1.—TENSILE TESTS.

drawn and cooled in a pit with many other filled pipes, and the whole covered with ashes. In 24 hours the pipes were emptied, the steel being at about 150° C.

For tensile testing the bars were turned down to a diameter of 0.564 inches, giving an area of 0.25 square inch. Through the kindness of Prof. I. A. Woolson, of the department of

TABLE IV.

No.	Ni.	C. C.	Gr. C.	Elastic Limit per Square Inch.		Ultimate Strength per Square Inch.		Elong. Per Cent. in 2 Inches.	Reduction of Area Per Cent.	Fracture.
				Tons.	Lbs.	Tons.	Lbs.			
KA*	3.79	0.41	21.68	48,563	40.28	90,246	26.0	44.75	Light grey granular; silky edges; half cup and cone.
KB*	3.79	0.41	20.74	46,469	31.94	71,559	28.0	49.4	Light grey granular; silky edges; half cup and cone.
LA	3.79	0.51	22.52	50,469	44.24	99,109	21.0	39.05	Light grey granular; silky edges; half cup and cone.
LB	3.79	0.51	18.66	41,812	34.75	77,861	25.5	46.78	Light grey granular; silky edges; half cup and cone.
MA	3.76	0.63	25.01	56,041	51.52	115,421	16.5	26.27	Finely crystalline, and serrated edges.
MB	3.76	0.63	18.83	42,209	39.0	87,360	23.5	41.45	Grey granular; tendency toward cup and cone.
NA	3.81	0.79	29.84	66,850	60.55	135,194	14.0	21.27	Finely crystalline, with serrated edges.
NB	3.81	0.79	17.79	39,875	41.21	92,313	17.5	30.11	Grey granular; crystalline near the edge.
PA	3.79	0.92	35.04	75,000	69.42	155,502	8.0	14.08	Crystalline; radial furrows; serrated edges.
PB	3.79	0.92	17.71	39,688	46.45	104,068	15.0	22.61	Fine crystalline; serrated edges.
OA	3.75	0.97	35.76	80,120	70.01	156,827	7.5	8.39	Fine crystalline; even surface.
OB	3.75	0.97	17.87	41,035	46.46	103,187	13.0	22.43	Crystalline, with granular center.
QA	3.81	1.24	45.44	102,030	75.31	168,697	3.5	3.20	Fine crystalline; very level fracture.
QB	3.81	1.24	18.38	41,186	50.74	113,673	11.0	17.21	Rather coarsely crystalline; little glistening flakes.
RA	3.82	1.21	0.31	20.25	65,530	65.01	145,642	10.5	13.9	Crystalline, but grey in color; serrated edges.
RB	3.82	0.50	1.02	17.34	38,847	46.84	104,934	14.5	20.96	Crystalline, but grey in color; serrated edges.
SA	3.92	0.93	0.71	30.18	67,605	60.59	135,734	11.5	17.86	Very dark grey granular; uneven surface.
SB	3.92	0.21	1.43	17.81	39,910	20.34	65,732	27.5	46.89	Very dark grey fracture; almost cup and cone.
TA	3.79	0.91	0.91	29.66	66,440	58.76	131,612	12.0	13.92	Very dark grey granular; edges slightly silky, and deeply serrated.

*"A" after the number of a steel signifies subjected to treatment A, while "B" signifies subjected to treatment B.

Number ¹	Ni.	C. C.	Gr. C.	Elastic Limit.	Ultimate Strength.	E. Per Cent. in 2 Inches.	Reduction of Area, Per Cent.
T Rolled.....	3.79	1.03	.80	86,500	143,000	8.4	15.0
T B.....	3.79	.10	1.73	40,000	58,500	33.0	52.0

mechanical engineering, Columbia University, the testing was carried out in his laboratory. The results are given in Table IV., and also diagrammatically in Fig. 1.

The hardness, by Brinell's test, was decreased from 271 to 131.

He also subjected the steels to the compression test, using

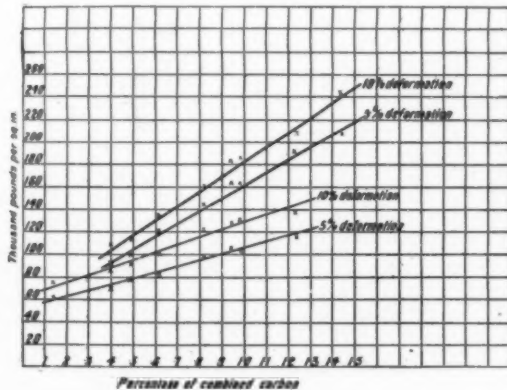


FIG. 2.—COMPRESSION TESTS.

test pieces 1 inch in diameter by 2 inches high. With the maximum pressure of the machine, 200,000 pounds, the pieces failed by bending, with the exception of the R rolled. All the pieces stood at least 20 per cent compression before bending, and in Fig. 2 is shown the load necessary to produce 5 per cent and 10 per cent reduction in length with the rolled steels, and those given treatment "B." The result is practically a straight line.

(To be concluded.)

Obituary.—Dr. T. C. Beilstein, professor of chemistry in the Institute of Technology of St. Petersburg, died on Oct. 19, at the age of 68 years. He is best known through his work in organic chemistry, especially on the aromatic series.

¹ Results obtained by Dr. Mathews on "T."

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

THE BRITISH ASSOCIATION MEETING.

Sir John Wolfe Barry's paper on "Standardization in British Engineering Practice" had its distinct metallurgical interest, as the engineering standards committee had been first concerned with rolled sections. The committee have laid down 16 sizes of equal angles, 30 of unequal angles, 20 of bulb angles, 6 of bulb tees, 7 of bulb plates, 8 of Z bars, 27 of channels, 30 of beams, and 20 of tees. Besides the above there are, advancing by 5 pounds at a time, nine sizes of bull-headed rails from 60 pounds to 100 pounds per yard, seventeen sizes of flat-bottomed rails from 20 to 100 pounds per yard, and five sizes of tramway rails 90 pounds to 110 pounds per yard, with their corresponding sections having a wider groove for use on curves.

As to whether standardization would bring with it disadvantages neutralizing its manifest advantages, it was pointed out that, should standardization be pushed too far it might in some instances stultify design or retard progress of invention; but provided that it is clearly understood that special circumstances must exist which warrant special designs, and as long as standardization is confined to broad principles, and the standards laid down are sufficiently numerous, the advantages must immeasurably outweigh the disadvantages. Exceptional cases may, no doubt, exist where the use of some particular form of angle or channel is justified, and where the expense entailed in cutting special rolls and delay in manufacture by the consequent changing of rolls is warranted; but in the vast majority of cases an engineer can equally well use standard sizes without in any way cramping or impairing his design.

In the case of steel standard sizes are also very valuable to trade, as they avoid the necessity of cutting new rolls, they can be rolled to stock, thus obviating any interruption in process of manufacture or delay in delivery. As a practical illustration the testimony of a large steel maker in Scotland may be cited, who admits that since the introduction of standard sizes his firm has been able to break up some hundreds of tons of rolls, and by no means the least advantage gained is that in his works the process of manufacture is now no longer so con-

stantly interrupted as it used to be, due to the frequent changing of the rolls to produce in smaller quantities the many special sizes asked for, without any corresponding advantage to the consumer.

The next paper was by Mr. W. Rosenham, on "The Deformation and Fracture in Iron and Steel," in which the author described his method of studying metallic fractures by means of transverse sections, the accuracy of such sections being secured by first embedding the specimen in electro-deposited copper and then cutting through the compound mass. Sections of various types of fractures were shown, and the general conclusion arrived at that in mild steel, fractures occurring after considerable plastic deformation pass indifferently through both ferrite and pearlite areas, while fractures due to shock, or otherwise occurring without much distortion of the metal, pass almost exclusively through ferrite areas. In conclusion, the author showed how his method could be used to study the mode of action of cutting tools.

A paper by Mr. J. E. Stead on "Segregation in Steel Ingots and its Effect in Modifying the Mechanical Properties of Steel," was then read, in which it was pointed out that the part of an ingot which cooled first was the purest, and that segregation consisted in the transport of impure matter mostly towards the upper central axis of the ingot, and also in the central cavities. The chief elements segregating out were carbon, phosphorus and sulphur in the relative ratios of 1:1.5:2, sulphur segregating most and carbon least. Gases and blow holes assisted segregation. For limiting segregation fluid compression of the ingots in the mould was the safest remedy, but part of the segregation could be reduced by the judicious use of aluminium or silicon.

The vagaries exhibited by the nickel heating coil of an electrically-heated porcelain-tube furnace afforded to Dr. H. C. H. Carpenter the subject matter of a paper on "Structural Changes in Nickel Wire at High Temperatures," and described the fundamental changes in the mechanical properties of such wires. The wire contained 98.60 per cent nickel, 1.22 per cent iron, 0.16 per cent manganese, and a trace of cobalt. Some dissolved gas or gases were also present. The diameter of the wire was 1/16th of an inch. The ultimate tensile stress was 35.2 tons per square inch, with a percentage elongation of 34.4 on 3 3/4 inch, and a percentage reduction of area of about 70. The resistivity at 0° C. was 9.2 microns-cm. In actual use the wire carries 20 amps. at 50 volts pressure, and a temperature of 1,200° to 1,300° C. can be obtained in the tube.

With care the life of such a furnace is usually three to even more months. But sooner or later it breaks down. The wire is then usually found to be so brittle that it can be snapped between the fingers. Occasionally it is still tough, but has become perfectly fibrous. These changes of mechanical properties are accompanied by structural changes which have been studied with the microscope. They are the result of the combined influence of heat and electricity, and are not produced by either of these agencies singly. It appears that the changes are due mainly to two effects, viz.: recrystallization and the penetration of gases, which are themselves the result of heat and electricity on the metal. The frequent association of brittleness with gross crystallization has long been known. But the evolution of dissolved or combined gas or gases from nickel and their mode of penetration through an eventual exit from it by means of cracks between the gross crystals are, it is thought, described here for the first time.

The last metallurgical paper to which I must refer was that by Mr. W. Taylor, on "A Magnetic Indicator of Temperature for Hardening Steel," in which the author described experiments with a magnetic induction balance used for ascertaining the critical temperature in the heating of steel for hardening. Also a simple magnetic attachment to a muffle furnace, used for giving an audible signal when the steel heated in the muffle has reached its critical temperature. He took advantage of the change of magnetic permeability at the critical point; when the

steel attains this temperature it becomes non-magnetic, and the sound in the telephone ceases abruptly; the steel is then quenched and hardened. The hardening in all cases investigated had been excellent, and not a single piece had been lost by cracking in the hardening. This is in fact a form of pyrometer with one—a predetermined—point of indication, but requiring continuity of attention lest the operator finds too late that the steel has lost its permeability, by which time the steel may have become too hot for quenching.

Before passing on to the work of other societies I must refer to a highly controversial paper read before another section by Mr. Swinburne, on a physical subject, "Radiation from Gas Mantles," in which their relation to radiation of selective emission, luminescence, catalytic action, resonance and unstable oxidation were discussed. On the theory of resonance, Mr. Swinburne said: "Particles of gas are vibrating and changing their velocity fast enough to produce light, but somehow do not produce it. Particles of solid, however, get into vibration synchronously with the gas particles, and thus radiate energy of the same grade as the heat of the flame. But ceria is supposed to be specially timed to vibrate with frequencies corresponding to visible radiation; so it radiates more lights than other solids. This really amounts to ceria doing the work of Maxwell's demon, except that it is working on waves instead of particles."

The concluding words were characteristic of the author: "The explanation that there is nothing anomalous about the mantle, and that it gives light just because it is very hot, has a Jordan simplicity about it which makes it unpopular compared with Abana and Pharpar of luminescence and catalytic action. But all the same, the very simple explanations are often wrong."

METALLURGICAL WORK AT THE NATIONAL PHYSICAL LABORATORY

Among the papers put aside for subsequent use during a summer overcrowded with copy, was that of the report of the National Physical Laboratory for 1905, a very lengthy and detailed document. I can only refer to some of the points of chief electrochemical and metallurgical interest.

In the Thermometric Division, Dr. Harker extended his work on high temperature furnaces; his application of the cascade principle of heating to these furnaces has proved very fertile. The small furnaces first constructed were described in a paper read early in the year before the Royal Society. (Our Vol. III., p. 273.) These led to a redetermination of the temperature of the melting-point platinum, which was found to be 1,710° C. \pm 5° C. Some experiments by Profs. Holborn and Henning, at the Reichsanstalt, published in 1905, give the figure 1,710°. Thus it seems clear that the value 1,780°, usually accepted, is 70° too high. During the year Dr. Harker also published a research on the specific heat of iron up to 1,100° C. (our Vol. III., p. 436), while another investigation, which is now practically ready for publication, is a comparison of the Kew mercury thermometer scale and the hydrogen scale, between 0° C. and 100° C.

In the Metallurgical Division, the nickel-steel research of Dr. Carpenter, Mr. Longmuir and Mr. Hadfield, was concluded during the early part of the year. It was presented to the alloys research committee in April, and read before the Institution of Mechanical Engineers at the end of the year. The committee of the laboratory were indebted to the institution for a continuation of their grant of £200, which has been employed in accordance with the wishes of the Alloys Research Committee in an investigation into the properties of the aluminium-copper alloys. Good progress has been made with this, but this has only been possible on account of the help of the Broughton Copper Co. and the British Aluminium Co., who have provided all the raw materials and given every facility for the forging and rolling of the alloys. These have been made by Dr. Carpenter and Mr. Edwards at the laboratory.

In addition to the research work in connection with the

aluminium-copper alloys, to which reference has already been made, a program was outlined in connection with "An International Steel Research." This reads as follows: "Steels of carbon percentages about 1.4, 0.9 and 0.6 carbon will be examined. Two series will be tested. In the preparation of the one, aluminium has been used; in the other it has not. Polished specimens will be heated for various periods at 1,000°-1,200° under conditions which prevent oxidation and afterwards quenched in a suitable bath. The specimens will then be microscopically examined with no further treatment other than possibly etching. To characterize the austenitic structure two tests will be employed—those of chemical attack and polish. The conditions will be studied under which the austenitic structure is destroyed by heating at definite temperatures and during a definite time period. Subsequently it is hoped to examine the conditions of production of the troostosorbic structural types, and for this purpose two more steels will be added to the series with 0.2 per cent and 1.8 per cent carbon."

THE INSTITUTION OF MINING AND METALLURGY.

The gold medal of the institution has been awarded to Prof. Bauerman "in recognition of his services in the advancement

of metallurgical science." The honor is one which is well deserved, and the recipient's many friends are cordial in their felicitations.

MARKET PRICES DURING SEPTEMBER.

The chief feature during the month was the phenomenal position of copper. Rising from the abnormal price of £84.15 per ton, the price of £90 was reached on Sept. 24, and £92.5 by the end of the month. Tin has been somewhat irregular, closing at £190, the lowest price of the month being £182 on Sept. 17. Lead has risen steadily from £17.10 to £19. Zinc is quoted at £32.15. Cleveland pig iron has been firm throughout, the highest price being £2.14.9 per ton, and the lowest £2.14.3. This does not mean stagnation, but an adjustment of supply to the present heavy demand, which is somewhat unusual. As for platinum, to pass from one of the commonest to one of the rarest metals, the price of one hundred shillings was paid at the end of the month for scrap platinum. Copper sulphate has, of course, risen in sympathy with the high copper prices to £26. Shellac is fetching £11 per cwt. No changes call for special comment in regard to the alkali trade or to the coal-tar derivations.

LONDON, Oct. 6.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

Copper.

Pyrite Smelting Without Coke.—The third installment of this lengthy and interesting article by Mr. L. T. Wright in the *Mining and Scientific Press*, Sept. 29, is devoted to thermochemical considerations. The author discusses in particular the advantages accruing from the use of hot blast, and gives some interesting comparisons, based upon results obtained in practice, averaging several monthly runs in which the sulphur contents of the charge was much the same for a set. The relations between sulphur and coke, using cold blast, is expressed in the following table:

	S in Fe S %	S in Fe S ₂ %	Total Sulphur %	Coke %	Cu. in Matte %	Furnace Months.
A	13.5	2.2	15.7	8.92	34.8	4
B	11.7	6.8	18.5	8.55	29.4	6
C	12.8	7.1	19.9	8.43	27.9	4
D	7.5	17.5	25.0	7.88	22.8	4

All the sulphur in the unroasted ore is calculated as FeS₂, and all that of the roasted ore, slag, flue dust, etc., as FeS. The results are calculated to unit weights of sulphur and coke per 100 unit weights of the actual average charge. It appears from the table that the replacement of roasted pyrite by the raw pyrite is attended by an increase in the total sulphur, a diminution of coke, and a lower grade of matte, 70 per cent of the total sulphur being oxidized to SO₂. Comparing the extreme numbers of the table it is found that 1 unit weight of sulphur oxidized replaced 0.159 part of coke. Theoretically, from the heats of combustion, if it had been oxidized by the reaction $\text{FeS} + 3\text{O} = 113,000 \text{ Cal.}$, it should have replaced 0.53 part coke, containing 83 per cent of carbon. If it had been oxidized by the reaction $\text{FeS} + 3\text{Fe}_2\text{O}_3 = 85.2 \text{ Cal.}$, it should have replaced 0.4 part of coke. On comparing the results A and C in the table a difference of one unit weight of sulphur oxidized is seen to have replaced 0.166 part of coke instead of 0.53 or 0.4 part respectively. On comparing A and B it appears that one unit of sulphur replaced 0.188 part coke. A similar comparison between the sulphur and coke in the case of warm blast is given in the following table:

	S in Fe S	S in Fe S ₂	Total S	Coke	Matte % Cu.	Blast 24 Hrs.	Rate of Smelt'g Tons per Sq. Fl. per Hrs.
E	5.51	15.16	20.67	4.0	31.0	28.4	8.58
F	7.20	12.60	19.80	4.5	31.6	28.6	8.41
G	7.15	11.30	18.45	4.8	31.4	28.4	8.47

These results have been obtained with warm blast in three furnaces similar in design and operation, each result being the average of six months' work with the same furnace. The three furnaces are stated to have been working under the same conditions, and to have exhibited much greater regularity in respect to coke consumption day by day than was shown by the same furnaces when they were working previously under cold blast. As in the case of cold blast, the coke consumption diminishes with addition of sulphur to the charge, special attention being given to reducing the coke consumption to a minimum. The average rate of smelting shows that the furnaces ran with great regularity and very fast. The percentage of sulphur oxidized to SO₂ is calculated as 71 per cent. A difference of one unit weight of sulphur oxidized, comparing the extreme results, is equal to a difference of 0.508 part of coke. This result shows quite a contrast to those obtained with cold blast, and seems to the author to illustrate again the value of the warm blast, which insures more active and regular smelting of the charge by increasing the temperature in the smelting zone. The author also makes a calculation to ascertain the vanishing point of carbonaceous fuel on a pyrite charge in order to find out how much additional sulphur should replace the 4 per cent coke used when 20.67 units of sulphur existed in 100 units of charge and 14.67 units of sulphur were oxidized, or how much sulphur must be oxidized per 100 units of charge to enable the coke to be entirely eliminated. From the extremes in the table given above he calculates that an additional 7.86 units of sulphur oxidized would, if the same relation between sulphur and coke continued, be sufficient to replace all the coke. Thus other things being equal, $14.67 + 7.87 = 22.54$ of sulphur oxidized per 100 units of charge, should enable the charge to be self-smelting. This the author later found to

be possible, and often the furnaces ran with just that amount of oxidation. A little excess, say 25 units of sulphur per 100 of charge, is, of course, better. The author devotes the latter part of the paper to the discussion of a heat-balance sheet for pyrite smelting per unit charge, and concludes by remarking that the use of warm blast is a logical sequence of a theoretical examination of the heat and the other reactions involved, and that these in their turn explain our practical experience that, without coke, and with cold blast, pyrite is not self-smelting.

A Hot-Blast Stove Efficiency Test.—The matter of using cold or hot blast in the operations of pyrite smelting is one of considerable interest, and opinions pro and con have been frequently expressed. Recently the consensus of opinion seems to tend towards the adoption of the hot blast. In the *Mining and Scientific Press*, Sept. 15, 1906, Mr. Herbert Haas gives an account of an efficiency test made in a U-type of stove of his own construction, at the blast furnace plant in the Afterthought Smelter, at Ingot, Shasta County, Cal. During the months of March to July, 1905, the working conditions of the stove were determined by careful records of the average volume of air heated per 24 hours, the average air pressures at the blowers, inlet and outlet of hot-blast stove and tuyeres, the average temperature of the inflowing and outflowing air, the average temperature and composition of the combustion gases at the stove-flue outlet and the average heat value and quantity of the wood burnt. He finds the following heat balance for the stove: Heat furnished by 8.5 cords wood: 2,800 pounds \times 60 per cent = 1,680 pounds; 1,680 pounds \times 0.4536 kg. \times 8.5 cords = 6,477 kg.; 6,477 kg. \times 3,800 kg. Cal. = 24,614,120 kg. Cal. Heat used, air: 7,500 cub. ft. \times 0.028317 cub. met. = 212.4 cub. met. min.; 212.4 cub. met. min. \times 0.2375 (284° — 19° C.) \times 1.440 min. = 12,247,904 kg. Cal. Water in wood: 2,800 pounds \times 40 per cent \times 8.5 cords = 9,520 pounds; 9,520 pounds \times 0.4536 = 4,318.3 kg. H₂O. Regnault's formula: W = heat units (kg. Cal.) required for raising 1 kg. water at t° C. to saturated steam at T° C, 606.5 — 305 (T° — t° C.) = 606.5 — 305 (200° — 19° C.) \times 4,318.3 = 661.7 \times 4,318.3 = 2,857,419 kg. Cal. Combustion gases at 200° C. computed from analysis of gases and wood substances (cellulose), which contained 44.4 per cent carbon, 49.4 per cent oxygen, 6.2 per cent hydrogen, 24,614 kg. nitrogen \times 9.2438 \times 200° C. = 1,200,178 kg. Cal.; 3,451 kg. carbon dioxide \times 0.2169 \times 200° C. = 149,704 kg. Cal.; 3,200 kg. water \times 0.475 \times 200° C. = 304,000 kg. Cal. Thus: nitrogen 1,200,178, carbon dioxide 149,704 and water 304,000, a total of 1,653,882 kg. Cal. The excess of air supplied to the fire-box carried away 618,500 kg. Cal. As tabulated in the following table:

Air	19,247,904 kg. Cal.	= 78.2% of A
Water	2,857,419 "	11.6 "
Combustion gases	1,653,882 "	6.7 "
Excess of air	618,500 "	2.5 "
Radiation, etc., to balance..	236,415 "	1.0 "
<hr/>		
24,614,120 kg. Cal. = A =	24,614,120 "	100.0 "

These results show that 78.2 per cent of the heat value of the wood was transmitted to the blast furnace in the form of heated air. In the opinion of the author the excess of water in the wood is responsible for the low figure, as in his opinion with coal or oil fuel the absolute efficiency would probably exceed 85 per cent. The 19,247,904 kg. Cal. would equal 2,406 kg. coke with a calorific value of 8,000 kg. Cal., or 2.65 tons at 2,000 pounds. The price of coke delivered at the smelter was \$21.00 per ton or, for 2.65 tons, \$55.65. The operation of the hot-blast stove cost \$38.00 per 24 hours, being \$17.65 in favor of hot blast.

Ore Conveying at the Blast Furnaces at the Cananea Copper Co.—In the course of an illustrated article on the La Cananea Mining Camp in the *Engineering and Mining*

Journal, Oct. 6, 1906, Mr. Dwight E. Woodbury gives a description of a new application of the conveying belt system for bedding the ores and charging them into the blast furnace. The plant consists essentially of two systems of rubber belt conveyors and trippers. One belt system is used for taking the ore from the hopper-bottom bins, into which it is received from the railway dump cars, and passing it through the sampling mill. There the coarser lumps are crushed and an accurate sample is taken. The conveyor then stacks the crushed and sampled ore on long ore beds of triangular cross-section, which are built up by a tripper, which moves back and forth on an overhead belt over the center line of each ore pile. In its shuttle-like passage back and forth it drops ore always on the crest of the pile. Successive additions of ore are thus disposed in concentric layers, thus giving a uniform composition for all cross-sections from one end of the pile to the other. Three such stacking conveyors are provided, each for a pile of 8,000 tons, a total bedding capacity of 24,000 tons. The second system of belts has been installed for conveying the smelting mixture from the reclaiming machine, as it attacks the pile, to the feeding devices at the blast furnaces. A conveyor belt passes along a trench which runs on one side of each ore pile. The reclaiming machine is a great girder-like car, which spans ore floor and trench, and, as it moves longitudinally towards the ore pile to be reclaimed, it cuts a slice clear across the bottom of the pile by means of a scraper-conveyor. It then delivers the material in a continuous stream into a hopper which feeds the belt-conveyor in the trench. A rake-like tickler keeps the end face of the pile sloping towards the machine, so as to present a thin edge for the plow blades of the scraper conveyor, and also prevents undercutting and caving. The carriage is actuated by electric motors, the speed of cutting and advance being readily regulated by the operator, who is guided by signals automatically set at a semaphore bin near the furnaces. The semaphore bin is placed between two of the belts in the series leading to the furnaces, and is supported on springs which allow a few inches of vertical movement, depending on the weight of ore contained in the bin, and thus actuating the semaphore signal arm up or down, according as the weight of the ore decreases or increases. The furnaces will be equipped with a mechanical feeding device, consisting of a long, narrow box on each side, of a size to hold a furnace charge. This can be set in a vertical position to be filled from the conveyor belt and then swung down to a horizontal position opposite each furnace door, whence the charges are dropped into the furnace. The old-style furnaces will be served by dropping the charges in front of the feed doors from overhead measuring boxes filled by the conveyors. The charges are then to be fed into the furnaces by hand. The ore-feeding system is thus based on using measured and not weighed charges. As, however, the measuring boxes can be checked against the known weight of the ore bed, it is expected to keep well within allowable limits. It is stated that it has been found possible during the last year or two to reduce materially the amount of barren flux introduced into the furnaces, so that the concentrates fed in now run about 9.5 per cent copper, 22.8 per cent silica and alumina, 28.5 per cent iron, 31.8 per cent sulphur, and a small amount of silver. Raw ores running above 5 per cent are smelted directly. The great bulk of the ore smelted is concentrates.

The Kiddie Hot-Blast System for Copper Smelting Furnaces.—The advantage of hot blast for copper-smelting furnaces seems to be pretty well established, but the cost of obtaining it in hot-blast stoves has militated against its introduction. A system in use at the Tyee Copper Co. Smelting Works, at Ladysmith, Vancouver Island, B. C., invented by Mr. T. Kiddie, and described by E. Jacobs in *Engineering and Mining Journal*, Sept. 29, depends upon utilizing the waste heat from the blast furnaces, by running a steel pipe, 6 inches in diameter, the entire length of the dust chamber, and return.

The pipe is designed so as to give the greatest heating surface and at the same time keep its cross-section down to the lowest practical limit. Its shape is such that the distance through which the heat must radiate to its center is reduced to a minimum, that is, approximately 4 inches in a circular pipe of equal area. As far as the cost of installation is concerned the author claims that it is approximately 50 per cent less than with the ordinary hot-blast U-pipe stove, and that it can be applied to any modern copper-blast furnace without disturbing the brick work of the furnace above the feed floor. On account of the comparatively low temperature, namely, 400° to 600° F. of the gases passing through the dust chamber, and the long travel and small cross-section of area of the heating pipe, the wear and tear is reduced to the lowest limit. In the arrangement of the pipe the lowest part of the double horizontal flue leading from the dust chamber to and over the furnace is kept sufficiently high above the charge in the furnace to prevent any burning of the steel plates. The outer jacket, which is in sections, can be readily taken apart if necessary, and all the joints are equally accessible. The author gives the following results obtained by three months' hot blast practice, in order to show the saving of coke, and the superior oxidizing power of the installation under consideration as compared to cold blast under similar furnace conditions: Running with hot blast—The charge consisted of 37 per cent roasted ore, containing 6.32 per cent sulphur and 4.08 per cent copper; 63 per cent raw ore, containing 16 per cent sulphur and 4.08 per cent copper; coke used, calculated on the ore smelter, 9.36 per cent; coke used, calculated on the mixture smelted, 7.70 per cent. The matte produced contained 42 per cent copper, sulphur to copper, 3.04 sulphur to 1 copper. Running with cold blast—The charge consisted of 63.62 per cent roasted ore, containing 6.32 per cent sulphur and 4.08 per cent copper; and 36.38 per cent raw ore, containing 16 per cent sulphur and 4.08 per cent copper; coke used, calculated on the ore smelter, 13.93 per cent; coke used, calculated on the mixture smelted, 12.97 per cent; matte produced contained 42 per cent copper, sulphur to copper, 2.42 sulphur to 1 copper. Thus it is seen that, as compared with cold blast practice, the hot blast made a saving of 33 per cent coke on ore smelted and 41 per cent on charge, the tonnage smelted being practically the same. In regard to oxidation, the respective percentages of roasted and raw sulphide ore used in the furnace charges were reversed, the hot blast producing equally good results with a two-thirds proportion of raw ore as the cold blast with a one-third, the product being in both cases a 42 per cent matte. The saving in ore-roasting cost therefore represented about 50 per cent.

LEAD.

Lime Roasting of Galena.—Within the last few years the roasting of galena by the old methods is gradually being abandoned in this country and abroad, and other processes, chief of which and most widely used at the present time, is the Huntington-Heberlein process, are taking their place. A paper by Mr. W. R. Ingalls, presented before the London meeting of the American Institute of Mining Engineers, and published in the *Bi-Monthly Bulletin* of the Institute, September, gives a concise review of the various processes practiced, but deals more extensively with the Huntington-Heberlein process and the economies to be effected by it. In the practical operation of the process the ore, limestone and silica are crushed to pass a 4-mesh screen and are then charged into a circular roasting furnace with revolving hearth. The diameter of the latter is 26 feet, and it roasts about 80,000 pounds of charge per 24 hours. The consumption of coal for reducing the sulphur of an ore containing 20 to 22 per cent down to 10 to 11 per cent is about 22.5 per cent of the weight of the charge. Two furnaces can be managed by one man per 8-hour shift. On the basis of the tonnage assumed above the cost of roasting is approximately estimated as follows: Labor, three men at \$2.50 = \$7.50; coal, 18 tons at \$2.00 = \$36.00; power, \$3.35;

repairs, \$3.35; a total of \$50.20 for 80 tons, or 63 cents per ton. From the roasters the ore is charged into the hemispherical converters of cast iron, 9 feet in diameter at the top, and about 4 feet deep. A perforated grate prevents the charge from falling to the bottom and stopping up the air connections. Two roasting furnaces and six converters are stated to be rated nominally as a 90-ton plant. This rating, however, is considerably in excess of the actual capacity, at least on certain ores. The operation of blowing usually requires from 12 to 16 hours, and at the end the charge is dumped by inverting the pot. The ordinary charge for the standard converter is stated to be about 8 tons of ore, weighing about 166 pounds per cubic foot. The finished charge contains from 3 to 5 per cent of sulphur. Each converter requires 400 cubic feet of air per minute. The cost of converting is approximately estimated as follows: Three foremen at \$3.20 = \$9.60; 9 men at \$2.50 = \$22.50; power, 21 hp., \$6.30; further supplies, repairs and renewals, \$5.00; a total of \$43.40 = 60 cents per ton of charge. As the cost of converting is reduced directly as the time required for the operation, the above estimate is based upon unfavorable conditions concerning the time required for working a charge. The total cost of treatment, from the initial state to the delivery of the desulphurized ore to the blast furnace is estimated as follows, per 2,000 pounds of charge: Crushing 1 ton at 10 cents = \$0.10; mixing 1 ton at 10 cents = \$0.10; roasting 1 ton at 63 cents = \$0.63; delivering 1.1 ton to the converter at 12 cents = \$0.13; converting 1.1 ton at 60 = \$0.66; breaking 0.9 tons at 60 cents = 0.54; a great total of \$2.50. The increase in the capacity of the blast furnace is attributed by the author to three things, namely: 1. Delivering to the furnace a charge containing a reduced percentage of fine ore, the speed of the furnace is increased, that is, more tons of ore can be smelted per square foot of hearth area; 2, there is less roasted matte to go into the charge; 3, under some conditions the percentage of lead in the charge can be increased, thus reducing the quantity of gangue that must be fluxed. The last part of the paper is devoted to notes on the operation of the Carmichael-Bradford and the Savelsberg processes.

ZINC.

Composite Retorts for the Manufacture of Zinc.—It is a well-known fact that the retorts in the manufacture of zinc are a source of considerable trouble. A good deal has been accomplished of late years by the introduction of presses for manufacturing them, but for refractory ores high in iron and lead, of other ore of easily fusible character, not even the retorts manufactured by hydraulic means, are altogether satisfactory. Retorts may, of course, be made from a more refractory material than is ordinarily used, namely, graphite, chromite, carborundum, bauxite, etc., but they are very costly. In order to do away with part of the expense it has been proposed to line the retorts with materials such as dolomite or magnesite, which is made to adhere to the surface proper of the retort by the interposition of silicate of soda. Another process for the manufacture of so-called composite retorts has been elaborated by A. L. Queneau, and is described by him in the *Engineering and Mining Journal* of Oct. 13, 1906. The requirements for retorts in the zinc manufacture are summarized by him as follows: 1. High refractoriness to withstand temperatures that often reach 1,600° C. 2. Chemical composition adapted to resist the corrosive action of bases, such as FeO, MnO, CaO, MgO, etc., and of fusible sulphides and compounds, such as ferrous sulphide, fluorspar, garnet, etc. 3. Good heat conductivity, since the charge is heated only by transmission through the vessel's walls. 4. High tensile and compressive strength, together with the ability to resist the abrasive and impact blows of the charging and cleaning tools. 5. Dense walls, to be little penetrated by the metallic vapors. 6. High elasticity in order to endure the repeated contractions and expansions due to the varying temperatures. 7. Low cost. The composite retort constructed by the author is obtained in

one operation by means of the Dor hydraulic press. It has a main body portion of the usual fire-clay and sand mixture, and an outer surface of predetermined and suitable thickness, made of a mixture of fire-clay and of a basic or neutral material, which latter material takes the place of the customary sand or chamotte, either wholly or partly, according to the requirements of a particular case. In the operation of the Dor press, owing to its automatic action, particles of clay placed in a like position in the various wads of material operated upon, cover always the same path while the retort is being made, and always go to form the very same portion of the finished articles. Therefore, if the relative positions of the particles in the wad and the resulting retort are ascertained experimentally, it will become possible to substitute in the proper zone of the wad the clay and sand mixture by a suitable refractory mixture to form the desired protective coating in any portion of the vessels. In starting the manufacture of composite retorts an initial batch of fire-clay, sand and the selected refractory material is prepared, the batch being made sufficiently for the first day's work. The refractory mixture for the following days is obtained by using the press wastage as well as that from the hammering machine. To each barrel-load of this material an appropriate weight of fresh refractory material is added, and the whole is then thoroughly mixed and pugged. The author states that the same number of composite vessels is made as when the press is used for the manufacture of ordinary retorts, the extra cost over the ordinary retorts being only due to the difference of price between the fire-clay and the selected refractory material. This extra cost is stated to vary between 30 cents for a Ceylon graphite mixture and 15 cents for a carborundum-sand mixture, the thickness of the protective layer being usually $\frac{3}{8}$ inch. According to the author, in a test lasting several months, made at the Palmerton (Pa.) plant of the New Jersey Zinc Co., it was found that with a protective layer formed of a mixture of Ceylon graphite with fire-clay, an increase of life of the retorts of 33 per cent was obtained in treating Willemite ores of the following composition: 46 per cent zinc, 3 per cent FeO, 8 per cent MnO, 1.8 per cent MgO, 3.5 per cent CaO, 24 per cent SiO₂.

Magnetic Separation at Galena, Ill.—In a communication to the *Engineering and Mining Journal*, Sept. 15, 1906, the author describes the process of separation of low-grade zinc ore adopted at the plant of the Joplin Separating Co., at Galena, Ill. The ore, after being crushed and screened, goes to an $\frac{1}{4}$ -inch impact screen, the undersize of which passes to two Wilfley tables, on each of which Galena, pyrites and blende are separated, and a clean Galena product is made, but only a rough separation of pyrites from the blende can be effected. The oversize from the impact screen goes to a seven-compartment jig having 30 x 36-inch screens of the following sizes, beginning at the lead end: No. 1, 5-mesh; No. 2, 8-mesh; No. 3, 4-mesh; No. 4, 5-mesh; No. 5, 6-mesh; No. 6, 7-mesh. The tailings flow direct into the creek, while a clean lead concentrate is made on the first two cells, and a mixed pyrites-blende concentrates on the other five cells. The concentrates are drained and roasted in a roasting cylinder, 32.5 feet long and 5 feet in diameter, with a capacity of 40 tons in 24 hours. The ore is roasted for $2\frac{1}{2}$ to 3 hours, the roasting being conducted in such a manner as to burn off one atom of the sulphur of the pyrites. The cooled ore then goes to a belt elevator, which delivers it to two trommels, each installed over a bin holding 30 tons. The product from either trommel can go to either bin. One of the trommels is fitted with a $\frac{1}{8}$ -inch perforated screen and also a $\frac{1}{4}$ -inch screen, while the other trommel has only an $\frac{1}{8}$ -inch screen. The oversize from the screens is crushed and returned. The product from the $\frac{1}{4}$ and $\frac{1}{8}$ -inch screens is kept separate, and each size is fed to a Cleveland-Knowles separator. The middlings product coming from the second magnets goes to 10 x 12-inch recrushing rolls, and then to a screen having 3 mm. holes, the oversize going back to the rolls again. The undersize goes to a third magnetic

separator, the middlings from which are not treated any further. Cutler-Hammer resistance boxes are used to regulate the amperage of the magnets. The voltage is kept at 90; the current used on the first or roughing magnets is 2 amps., and that on the finishing magnets is 6.5 amps. The average of the concentrates made last year are stated to have been for the coarse $\frac{1}{4}$ inch size, 60.49 per cent zinc and 2.11 per cent iron; for the fine $\frac{1}{8}$ -inch size, 57.07 zinc and 2.19 per cent iron. At the relative proportions of one part of the coarse to two parts of the fine material, the average was 58.21 per cent zinc and 2.16 per cent iron.

GOLD.

Cyanide Practice with the Moore Filter.—In the *Mining and Scientific Press* of Sept. 8, 1906, Mr. R. Gilman Brown concludes his lengthy article on the cyanide practice with the Moore filter at the Standard Mine, Bodie, Cal., the first part of which was abstracted in the October issue of *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*. The present part of the paper deals exclusively with the operations in the Moore filter plant. The essential difference between the Moore and the Cassel-Butters process is that Moore transfers the filters with their load of slime from the pulp vats to a wash-water tank, and then to the discharge hopper by means of a crane, whereas, in the Cassel-Butters process the pulp is removed from the filters by pumps of large volume, wash-water is substituted, and then discharged, the operation being conducted in the same vat. In the plant under consideration the filters are made of canvas of medium weight, 5 feet wide and 16 feet long, the filters being so constructed that percolation of the solution is possible. A frame of steel I-beams supports forty-nine of such filters hung 4 inches from center to center. This arrangement constitutes a so-called basket, which is approximately 16 feet square. On top of each basket rests a suction pump, driven by a 10-hp. d. c. motor. The filter space afforded by this arrangement is quite large, inasmuch as each unit comprises 7,840 square feet. The filtering vats are round and flat-bottomed, with a diameter of 24 feet and a depth of 7 feet. They are provided with four-arm agitators, which run close to the bottom at 7 r. p. m. The process is carried out as follows: A basket is lowered into the vat, which is full of pulp until the tops of the slats are submerged. The suction pump is then started, the first solution, which comes through muddy, being returned to the vat. Under ordinary circumstances the solution soon clears, but if there should be a leak in any of the filter plates, this can be quickly detected and the damaged filter cut out. The suction is continued with intermittent agitation until a sufficient thickness of deposit is obtained—in the present case an average of $\frac{3}{4}$ inch. The basket is then raised, the suction pump being still kept running, transferred by a crane to the wash-water tank, and lowered into the latter. The washing is continued with frequent titration of the filtrate until the solution has reached a predetermined minimum of strength in cyanide. About 0.7 tons of wash-water is stated to be needed per ton of dry slime. The cake in this condition carries about 40 per cent of moisture. The basket is then raised again and run over the discharge hopper, the suction being continued until the excess of moisture is removed. An air pressure of 35 pounds is then turned on in successive blasts of a few seconds each. This operation causes the cake of slime to drop off, the discharge averaging to 85 per cent of completeness. Every alternate day the filters are cleaned by substituting water under 20 pounds pressure for the air. The time consumed in this cycle of operation depends, of course, essentially of the thickness of the deposit, the average thickness of the latter for the last year in the plant under consideration being about 0.74 inch, and the maximum for one month 1.14 inches. The time of the accretion period is about $3\frac{1}{2}$ hours, washing and discharge consuming about the same time. Three complete cycles can thus be made in 24 hours with time for emergency matters. The author claims that this time is considerably longer than has been

experienced elsewhere, and that this fact must be attributed to the large percentage of clay in the materials. On the important question of the wear of the filters the author remarks that an extreme life of ten months has been noted in some cases, while six months can be taken as a fair average life. However, if they are badly made, or carelessly handled, they require constant attention, and half a cycle per day can easily be lost in this way.

Tube Milling in Korea.—An interesting experimental plant for carrying out experiments in tube milling with a cyanide solution, established at the Oriental Con. Mining Co., at Chit-talbie, Korea, is described in the communication by Mr. A. E. Drucker in the *Mining and Scientific Press*, Sept. 22, 1906. The plant consists of one tube mill, $2\frac{1}{2} \times 12\frac{1}{2}$ feet, two mechanical agitators with plow shoes, 8×6 feet in diameter, three filter and settling boxes, $4 \times 5 \times 5$ feet, forty-eight separate-compartment zinc boxes, two sumps and one stock solution vat. The method is continuous, the concentrate being put into the hopper with cyanide solution, nearly all being ground to slime in passing through the tube mill. A good percentage of the gold is claimed to be extracted. At the end of the mill is located a spitzkasten 3 inches wide by $1\frac{1}{2}$ foot long, by $1\frac{1}{2}$ foot deep, which is supplied with clear water from the bottom. The bottom discharge of the spitzkasten consists of coarse concentrates and clear water. What passes over the spitzkasten is the cyanide solution and slime. The coarse concentrate which may escape from the tube mill is caught in the settling boxes and again reground, while the overflow contains very little value, as it assays 6 cents per ton. The slime concentrates and the cyanide solution pass on to an agitator, which has been set in motion at 3 r. p. m., and the muller lowered. The agitation of the product is continued for about 15 hours, the muller being then raised 2 feet, the agitator stopped, and the concentrate allowed to settle. The clear cyanide solution is decanted into filter boxes, and from there passes through the zinc boxes. The strong precipitated cyanide solution from the sump is then pumped up to the agitator as a wash to partly remove the gold remaining within the settled concentrates. The muller is then allowed to work down on the charge and agitated for a few minutes, this being followed by settling and decantation. This operation is followed by two successive weak cyanide washes to remove the cyanide and gold, and finally a water wash is given which is run through a row of zinc boxes to waste. The charge of about 5 tons of clean concentrates is then discharged. The total treatment is said to consume 24 hours, both agitation and decantation being done in the agitators. While one agitator is being filled the other is decanting and getting ready to be discharged. Strong cyanide solution of 0.43 per cent is used in the tube mill and agitators; the weak solution which is employed for the washes has a strength of 0.1 per cent. No cyanide is added to the weak solution. About 2 pounds lime per ton is added with the concentrate as it is fed into the mill. The author states that the extraction obtained at present with the tube mill and agitators on clean concentrate is 93 per cent, and hopes to better these results by gradually making improvements.

Ore Treatment at the Smuggler Union Mill, Telluride, Col.—A description of the milling practice at the above plant is given in the *Mining Reporter*, Sept. 29, 1906. The Smuggler mill contains sixty 1,050-pound stamps, dropping 6 inches at a rate of from 100 to 108 drops per minute, crushing through 14-mesh No. 20 steel wire screens of 18×52 -inch size. The duty of each stamp is given as nearly 5 tons per 24 hours. The stamps are fed by simplified adjustable challenge feeders. The amalgamating plates are 8×4 feet. From the plates the pulp passes to stationary No. 24 brass wire screens of 20-mesh, the undersize dropping on bumping tables, and the oversize going to 5-foot Huntington mills, which are fitted with 20-mesh wire screens. The thirty-two 5-foot Triumph vanners set on a floor below are fed with battery pulp without classification. The vanner tailings are taken by a wheel-bucket elevator to the top

of the mill building, and are delivered by a launder to the cyanide plant. This wheel bucket elevator has an outside diameter of 28 feet with a rim of 12 inches, and is constructed entirely of iron. It is fitted with overlapping pockets or buckets, $5 \times 12 \times 18$ inches. As the wheel is rotated in the pit below, which contains the vanner tailings, the buckets are filled and, on assuming a horizontal position at the top, dumped directly into wooden receiving boxes, which empty into the cyanide plant delivery launders. It is stated that the operation of the wheel requires about 5 hp., and that during a continuous operation of twenty months, the repair expenses have been less than \$2.00. The tailings material from the concentrating plant goes to large settling tanks in the cyanide plant, the overflowing slimes being pumped back to the canvas plant while the slimes are treated by percolation. The canvas plant consists of ninety-eight tables, 10×12 feet, the overflow being retailed over four Wilfley slime tables, and the heavy material as washed down being still further concentrated over vanners and Wilfley tables. The canvas slime plant produces from 10 to 12 tons monthly of high grade concentrates that carry about 17 per cent moisture. None of the concentrates are dried before shipment. The cyanide plant is equipped with sixteen wooden tanks of 40 feet diameter, 8 feet depth. The solution tanks are 12 feet deep and 40 feet in diameter. The strength of the cyanide solution used is variable, on account of the diversity of the leaner ores treated, the average, however, being given as 8 pounds cyanide to the ton of ore for the strong solution and 2.5 to 3.5 pounds for the weak solution. The diverse character of the ores also causes a variance of the leaching period of from twelve to sixteen days.

Ore Treatment at the Combination Mine, Goldfield, Nev.

—A lengthy description of the method adopted for treating the ore of the Combination Mines Co., as well as customs ores from the district, by Mr. F. R. Bosqui, in the *Mining and Scientific Press*, Oct. 13, contains a number of interesting data. The mill is equipped for treating oxidized and sulphide ore. The ore is crushed by a 10×16 Sturtevant roll-jaw crusher, after having been mixed with the required amount of lime, varying from 5 to 10 pounds per ton. The crusher breaks the ore to about $\frac{1}{2}$ -inch size. The ore is then elevated to the mill bins, there being four of these bins for the four units of five stamps each. Ten more stamps have recently been added for treating sulphide ore. The following account refers to a complete five-stamp system. Inside amalgamation is carried on by means of a curved plate screwed to the chuck block, the ore being crushed through a 12-mesh wire screen. A splash-plate is used outside the screen, from which plate the pulp falls to a lip-plate about 12 inches wide. The front edge of this plate is slightly bent down, giving the pulp a gentle drop to the apron-plate. There are three plates to each mortar, arranged in steps, giving an amalgamating surface 53 inches wide and about $12\frac{1}{2}$ feet long. The whole tray, by means of wheels and track, can be shifted during the clean-up of the battery. At the bottom of each tray there is a small cone hydraulic classifier, which separates the coarse mill pulp into two products, namely: 1, fine sand and slime, which passes to the outer discharge lip of the Bryan mill, and thence direct to the concentrators; 2, coarse sand, which passes to the Bryan mill for regrinding. The ore is extremely hard and tough, and is therefore crushed with 1,350-pound stamps, falling 100 times per minute with a 6-inch drop. The stamp duty is only $3\frac{1}{2}$ tons, using a 12-mesh screen. One of the 5-foot Bryan mills, running at half speed, takes all the coarse sand from ten stamps and crushes it through a No. 9 slotted screen, equivalent to 40-mesh. The final product of the Bryan mill is passed over two 6-foot vanners and two 6-foot Triumph tables. The pulp is then raised by two Frenier sand pumps to two sets of cone classifiers. The top cone is not a classifier, but is so adjusted that it sends a fairly uniform flow of pulp to the two smaller cones. The first rough classification is made in the small hydraulic cones, from which a stream of sand flows

direct to the sand vats. The overflow from these small cones, consisting of slime and fine sand, flows to the larger lower cone, where a closer classification is made, the stream from the bottom of this cone also flowing to the leaching vats. The overflow from the large lower cone passes to the slime settlers. The separation was not ideal though the recovery from the slime (over 95 per cent) has been very good, but a contemplated re-arrangement of the three cones is expected to improve the extraction from slime. From the cones the sand flows to four settling vats on each side of the mill. When the sand-settler is filled the surplus moisture is removed by a Gould vacuum pump, and the charge shoveled to the treatment vat below, where it is given an eight days' treatment with a 0.1 per cent and a 0.2 per cent cyanide solution. The slime is delivered to the center of a conical-bottom settler, provided with a rim overflow, two settlers being arranged at each side of the mill. Each settler is alternately allowed to fill and overflow for 12 hours, and allowed 12 hours for settling. The surplus water is carried off by a pipe decanter, the slime being left with about 50 per cent of moisture. Sufficient strong cyanide solution is added to the charge of slime to make a solution of from 0.15 to 0.2 per cent cyanide, and to give the pulp a consistency of three parts solution to one of slime. By means of a centrifugal pump the pulp is then transferred to an agitator with a steep cone bottom, and the same pump then agitates the slime by taking it from the bottom of the vat and throwing it back at the top, a supplementary agitation being given by means of a mechanical stirrer, revolving slowly. The agitation lasts from 12 to 15 hours, and the pulp is then dis-

charged into the slime reservoir, which is a vat of large capacity, provided with a mechanical stirrer. It is thence drawn as required for filtration in the Butters-Cassel filters. In the plant under consideration there are twenty-eight frames, 5 x 10 feet, set $4\frac{1}{2}$ inches apart in a box 10 feet square, with a pointed bottom inclined at an angle of 50° . The slime pulp is introduced at the point of the box, vacuum of 22 inches of mercury being maintained for about 20 minutes, during which time a cake, $\frac{3}{4}$ to 1 inch thick, is deposited on each side of the filter frame. The surplus pulp is then withdrawn to the slime reservoir, and a wash, consisting of a weak solution of cyanide, introduced. When the cakes are thoroughly washed the weak solution is withdrawn into its respective vat, and water is introduced until the frames are completely immersed. The object of this final water is to assist in removing the cakes. More water is introduced into the interior of the frames under a low head, and this water causes the cakes to drop off clean into the pointed bottom of the filter box. From there they are finally removed by sluicing. The whole requires about $3\frac{1}{2}$ hours, and about 9 tons of dry slime are treated at each charge; the plant has, therefore, a capacity of 63 tons. One man on each shift is required for the operation. The author states that the filter plant has required no repairs since it was first operated in February of this year. The cost of filtering exclusive of power has been reduced from \$0.96 per ton of slime in January, when filter pressing was used, to \$0.26 in May, when the above described system was in operation. In the treatment of the sulphide ore a 4 x 12 Abbé tube mill of the trunnion type is used.

RECENT METALLURGICAL PATENTS.

COPPER.

Converter.—A novel converter construction for use with copper matte has been patented by Herbert Haas (832,665, Oct. 9), the object being to secure a uniform consumption of the lining in forming the slag, and a reduction in the labor and power required. The converter is shown in Fig. 1. It is built in two parts, an upper part 1 and a lower part 2, which are secured together by means of the iron bands 5 and 6, which have flanges 3 and 4 secured together by bolts 7. The compressed air is introduced through the hollow shaft 14, and then passes through the transverse air conduit 17, which is joined to the short pipe 18 conducting to the wind-box. The converter is rotated by means of the pinion 22. The wind-box is built in two parts, the casing 30 having an annular concave flange 31, adapted to be connected with the bottom of the converter, and a flange 32 for supporting the bottom of the wind-box, and, secondly, the bottom 33 of the wind-box, which is a plate for supporting at suitable angles the tuyere direction pipes 34. The object of these pipes is to give a suitable direction to the apertures which are formed in the converter lining over the wind-box, in order to produce the best result. This construction of the wind-box with direction pipes arranged tangentially in a ring about a central axis is a special feature of the converter, and the effect of this construction is that the blast imparts to the molten matte a rotary motion. The reduced copper cannot collect in the bottom of the converter so long as the blast is blowing, for the fluid mass is forced by the tangentially rising air currents to travel around the entire converter lining, thereby evenly corroding the lining. The direction of the tuyeres is such that the issuing air develops a centrifugal action upon the matte, and the reduced copper being of greater specific gravity than the matte is thrown by the centrifugal force toward the converter walls as far as possible from the orifice of the tuyeres or conduits, and thus avoids the chilling action of the cold blast, but at the same time centrifugal force also acts upon the iron, which is

being constantly reduced from the sulphide, and which having a greater specific gravity is thrown against the converter walls, and being nascently oxidized as the sulphur is burned off combines with the silica in the lining to form a slag.

Lining Converters.—When the basic or silicious lining of a converter has been stamped in, it is necessary to dry the lining carefully. An improved and shortened process of drying has been patented by H. L. Charles (832,895, Oct. 9), of Butte, Mont. The tuyeres are first closed and molten slag, low grade molten copper or molten iron matte from converters or reverberatory furnaces is filled in. Converter slag is preferred, as it has a higher temperature than all other known slags. A few pieces of wood or coal are then put on top of the slag or matte, and a light fire kindled to keep it from chilling. In about an hour a shell of slag or matte forms all around inside of the lining. The first slag or matte is then poured off, and a fresh, hot layer of slag or matte is poured into the converter. This can remain indefinitely, as the heat from the first slag or matte penetrates the lining so far that the remaining moisture will be totally driven out in 3 hours, and the lining remains hot for 2 hours. A complete and uniform hardening of the lining is thus secured in a short time. In addition the slag shell formed on the inside is easier smelted than the cold, hard silica lining, making the converter run faster and raise heat quicker than is possible in the old way. The resulting slag from first corrosion will run higher in iron than with the old process.

Smelting Copper Matte.—W. Kemp (832,738, Oct. 9) patents the following process for smelting copper matte so as to produce black or metallic copper directly therefrom in a single continuous operation. The copper matte is charged into the furnace in a hard state, mixed with silica flux, and a burning flame of oil vapor, steam and air is continuously forced from burners into and through the mass of the matte at the bottom, the power of the flame being very great (approximating a temperature of $1,200^\circ \text{C.}$) due to the large excess of air in it.

The action of the flame is to successively reduce the lowermost zones of the matte to a molten condition, the mass of the matte above being at the same time subjected to a gradual roasting action upwardly. As each lowermost zone is reduced to the proper state of fusion, the copper and slag are drawn off and the next succeeding and highly roasted zone takes its place. The chemical action is said to be that the copper-iron

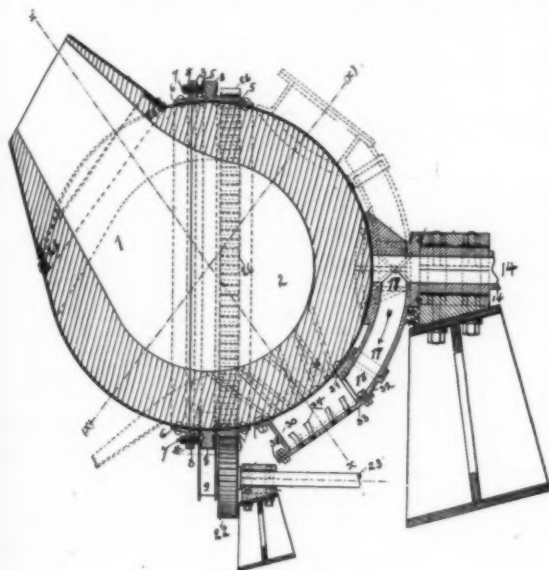


FIG. 1.—CONVERTER.

sulphide reacts with the excess of oxygen and forms metallic copper, ferrous oxide and sulphurous acid gas. The ferrous oxide then combines with the silica to form slag. The copper remains in molten metallic condition.

Welding Copper.—J. Spittall (832,755, Oct. 9) welds pieces of copper and other metals by first heating them to a cherry red. The parts of the metal which are to be made contiguous in the weld are then covered lightly with a composition consisting of 2 ounces of boracic acid and 1 ounce of phosphate of soda, thoroughly mingled by stirring. The two pieces to be welded together are then placed together, the composition not necessarily lying between them, and the welding is completed by hammering or pressure, as is done in the ordinary welding of iron.

NICKEL.

Orford Process.—We have repeatedly referred in these columns to the method of tops and bottom smelting of nickel-copper ores, used so extensively by the Orford Copper Co. The present practice is to smelt the matte containing sulphides of nickel, copper and iron in a cupola furnace, together with a charge of coke and sulphate of sodium, in the form of crude niter cake. The carbon of the coke reduces the sulphate of sodium to sulphide of sodium, and as this material when fused has a solvent action upon the copper and iron sulphides contained in the matte it will dissolve them to a considerable extent, so that when the molten contents of the furnace are tapped into settling pots the materials will stratify. Sulphide of nickel, together with some sulphide of sodium, will settle to the bottom, and the sulphide of sodium, with the copper and iron sulphides in solution, will float on the top. The mass when solidified is divided into tops and bottoms, the tops containing sulphides of copper, iron and sodium, and the bottoms containing sulphide of nickel, together with such of the sulphides of copper and iron as have not been taken up in solution in the sulphide of sodium. The bottoms are again charged into the smelting furnace with sulphate of sodium, and are resmelted a sufficient number of times to purify the material to

such an extent as to leave the nickel therein nearly free from copper, generally reducing the copper to less than 1 per cent. After having thus been treated the bottoms are crushed and leached for the purpose of removing the soluble portions of the sodium compound, and the product is dried and partially calcined, and is then mixed with sodium chloride and sodium sulphate and again calcined. The sodium chloride and sodium sulphate react to produce chlorine gas, which converts the copper contained in the mass partially to the form of copper chloride, and the remainder of the copper to copper sulphate, the nickel being converted largely to the form of nickel oxide and partially to the form of nickel sulphate. The calcined product is then leached with water for the purpose of removing the soluble salts of the three metals—namely, the copper chloride and copper sulphate, sodium sulphate and nickel sulphate. The residuum is then rewashed with dilute sulphuric acid, which removes the copper as a sulphate, but does not attack the nickel oxide. It has been the practice to precipitate the metals from these solutions by the addition of sodium carbonate or sodium sulphide and then to remove the precipitate by passing the liquor through filter presses, in which operation the sodium compound is lost.

R. R. Maffett, of the Orford Copper Co. (833,722, Oct. 16), has invented a modification of the last part of the process as described above. He avoids the precipitation and filtering. The liquor which results from the washing of the nickel-copper compounds with water and sulphuric acid, and which contains in solution sulphate of sodium and sulphates of copper and nickel, is withdrawn and introduced into an evaporating tank containing steam coils. There it is heated and concentrated to the point of crystallization, the sodium-copper-nickel-sulphate being recovered in the form of crystals. These crystals are taken from the vat, and can be introduced directly into the smelting furnace as a part of the charge of matte, coke and sodium sulphate in the same manner in which niter cake has heretofore been introduced, and it has the advantage not only of supplying the sulphate of sodium requisite for the smelting operation, but also of saving the contained copper and nickel, which pass with the molten contents of the furnace into the settling pots and are there recovered.

ZINC.

Distillation Process with Internal Heating.—The very low thermal efficiency of the zinc distillation process with a large number of retorts heated from the outside, is well known and well understood. It is just for this reason that attempts have been made to introduce the electric furnace in the zinc industry by using internal heating by means of the current and by using large-size furnaces. A process patented by H. Mehner (833,472, Oct. 16) also endeavors to use internal heating in larger furnaces, but without employing electric heating. His process consists essentially in bringing in contact with each other the zinc ore, the reducing agent, and an "incandescent fluid heat carrier of such a nature as not to absorb or otherwise do harm to the metallic vapors generated." The heat carrier may consist of a molten silicate (including slag, scoria, cinders and the like) heated above its melting point up to incandescence. The operation is indicated in Fig. 2; *h* is the furnace for heating the silicate, while *a* is a shaft furnace filled with coal, coke and the like. The zinc ore is added to the molten silicate, and is supplied to the top of the shaft furnace through the centrally located inlet *d*. The mixture of zinc vapors and carbon monoxide resulting from the reduction escapes from *m* into a condenser. The zinc vapors are obtained in as dense a state as in the usual distillation process, being only attenuated by such a quantity of carbon monoxide corresponds to $\text{ZnO} + \text{C} = \text{CO} + \text{Zn}$. On the other hand the great losses of zinc, which in the usual distillation process are caused by the permeable condition of the walls of the retorts, may be avoided by lining the furnace with bricks of dense carbon, which do not absorb zinc vapors or allow the vapors to penetrate them. The silicate escaping from the lower

part of the furnace through *f* is collected, heated anew to a bright cherry-red, at least, mixed with a fresh quantity of zinc ore and then used over again. In case the ore contains beside zinc other metals—such as lead, silver, copper, etc.—or is a so-called composite ore, then those metals will collect on the bottom of the furnace below the silicate, and the mixture may be tapped off through a tap hole at suitable intervals. The molten silicate may be replaced by a molten metal heated above its melting point up to incandescence. The author prefers to

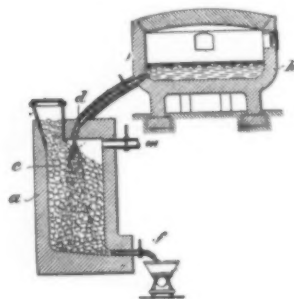


FIG. 2.—ZINC DISTILLATION.

use iron, since it is by itself a reducing agent for zinc ores, especially zinc blende. In case of sulphur-containing zinc ores, the iron, by its contact with the ore, drives out the zinc in the state of vapor and combines with the sulphur in the ore. The molten iron after being removed from the lower portion of the furnace is reheated by blowing a current of air through, whereby the sulphur in the iron is burned to sulphur dioxide. By the same process pure zinc sulphide may be produced instead of metallic zinc by producing sulphur vapor within the furnace, which combines with the zinc vapor and forms zinc sulphide, which is collected as an impalpable powder. To obtain pure zinc white, in a finely flocculent condition, oxygen or air, etc., is admitted to the escaping zinc vapors so as to burn them.

GOLD AND SILVER.

Treatment of Refractory Ores.—J. W. H. James (833,394, Oct. 16) patents the following treatment of refractory gold and silver ores, the object being to eliminate from the ore the contained relatively baser metals. In a cylindrical retort, heated from the outside by means of producer gas, the ore is roasted in a loose state without access of air, and simultaneously subjected to the action of water gas. The result of this treatment is that the ore becomes exceedingly friable so that it may be easily broken by hand and passed between crushing rolls. The gold particles are said to assume thereby a globular form, so that the loss in the form of float-gold is reduced to a minimum. The sulphur, arsenic, tellurium, antimony, bismuth, zinc, lead, iron, copper pyrites, iron oxides, and the other baser metals or materials with which the gold may be chemically or mechanically combined, are simultaneously eliminated from the ore. They are either carried off by the water gas as volatile compounds, or they are reduced to such a condition that the gold can be obtained by amalgamation or otherwise. The absorbed volatile compounds may then be readily separated by precipitation in suitable wet and dry chambers. The water gas is preferably employed in the condition of about equal volumes of associated but uncombined hydrogen and carbon monoxide.

Precipitation from Cyanide Solutions.—In ordinary cyanide practice the solutions are almost invariably alkaline in reaction. W. J. Sharwood (832,880, Oct. 9) first makes the solution neutral by adding either commercial sulphuric acid or acid sodium sulphate. If finely divided zinc is then used for precipitation, it is claimed that the amount of zinc dissolved during precipitation is greatly reduced, and that the precipitation of gold, silver and copper is more perfect.

Ozone in Cyanide Process.—In cyanidation, as practiced at present, the oxidation is generally effected by aerating in large, flat tanks, the solution being allowed to percolate. This requires constant agitation for quite a long period, so that the pulp will be exposed over and over again to the action of the air. F. J. Crane (827,620, July 31) recommends the use of ozone. He forces the solution of pulp, with cyanide added, to

the upper portion of a tank, from whence it passes through an ozone-containing chamber, the solution being constantly taken off from and returned to the tank, and in its passage entrains an additional supply of ozone, which by the action of the circulating pump is thoroughly beaten into the pulp. In treating refractory ores previously ground to pass through, say, one-eighth or one-quarter-mesh, pulverization may be effected by the impact of the discharge of the pump being directed against an arrester located above the ozone-containing chamber. Nothing is said concerning the cost.

IRON.

Treatment of Fine Ores.—A. J. Mason (833,406, Oct. 16) patents the treatment of fine iron ores in a furnace, illustrated in Fig. 3. The fuel is pulverized and introduced into the reducing and melting chamber 2 from the hopper 1 with the aid of a strong air blast. The finely-divided ore and flux are supplied through the hoppers 7 and 8, respectively, and drop in a finely-intermingled stream downwards through the stack 9. In the melting chamber 2 reduction takes place to metallic iron which flows in a molten stream through the discharge spout 15 into a pot underneath; 17 is a bed of coke, which constitutes a lining for a portion of the bottom and side of the melting chamber 2, against which the combustible gases and fuel are directed to complete the reduction and melting of the ore by furnishing an incandescent bed on and through which the partially reduced and melted ore drops down. This bed of incandescent coke will remain unconsumed as long as the pulverized fuel itself furnishes all the carbon required for reduction. The hot gases are used for regenerative heating. The essential differences between this process and blast furnace practice are, first, that all materials are introduced in finely-divided condition and in shallow broad streams, and, second, that the carbon for reduction is not introduced together with the charge at the top but is supplied directly to the reducing zone.

Briquetting.—H. Schulte-Steinberg (833,005, Oct. 9) uses for briquetting ores a binding material made from blast furnace slag. The comminuted slag, preferably white furnace slag, is exposed in a revolving cylinder to the quick and thorough action of high-pressure steam. The binder thus obtained is intimately mixed with the refuse or rotten ore in

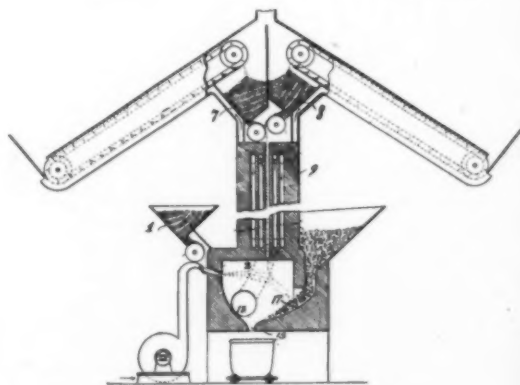


FIG. 3.—TREATMENT OF FINE ORES.

the proportion of 1 to 10, and then the mixture is applied to any kind of briquetting machine.

Iron from Copper Slags.—R. Baggaley (832,948, Oct. 9) endeavors to save the iron contained in the ferruginous slag of copper-smelting furnaces, and points to the fact that pig iron and steel are generally expensive and scarce wherever copper smelters are operated. The ferruginous slag discharged from copper-smelting furnaces is composed of silica combined with iron and other bases. This fact, of course, demands the use of fluxing agents. The silica is separated from the con-

tained iron by means of limestone as flux. The limestone combines with the silica to form a slag, and thereafter in the operation of bessemerizing the various impurities are eliminated in succession in the order of their oxidation. The last vestiges of sulphur in their elimination will demand the sacrifice of a considerable percentage of the iron, which is thought to be permissible in this process. In some cases also it will demand the almost complete elimination of the carbon, and in such cases the carbon must be restored to the bath by means of adding spiegeleisen or ferro-manganese, etc.

Purifying and Reheating Blast Furnace Gases.—D. Lamond and D. D. Lamond (833,467, Oct. 16) have patented the apparatus shown in Fig. 4, for utilizing the heat given up from the impure blast furnace gas, for reheating it after it is purified and before it passes to the furnaces, stoves, etc., for use as fuel. There are three large stove-like settling chambers, 2, 3, 4, connected in series by horizontal pipes 5, and having the usual bell and hopper-bottom construction 6 for the accumulation and discharge of impurities which separate by gravity from the gas. Each chamber is enclosed by an outer wall 7, thus forming a flue 8. The upper ends of the flues 8 surrounding chambers 3 and 4, are connected by elbow 10, while the lower ends of the flues surrounding chambers 2 and 3 are connected by duct 11. The upper portion of chamber 2 is connected by pipe 12 with the dust catcher 13 on the blast furnace. Chamber 4 is connected by duct 14 with the lower portion of the gas washer 15, the upper end of which is connected by the vertical pipe 16 with the lower end of flue 8 surrounding chamber 4. The blast furnace gas passes directly from the dust catcher 13 into the upper end of chamber 2, and then successively through pipes 5 into chambers 3 and 4. In passing through these relatively wide chambers the gas expands, and the heat is absorbed by the chamber walls, thereby heating the outside flues 8 to the temperature of the gas as it leaves the blast furnace. The passage of the gas through these chambers is comparatively slow, so that it can give off its heat. The gas then passes into washer 15, and in its upward passage it encounters downward sprays of water discharged from transverse perforated pipe 17, whereby all the dust and impurities are removed. The pure and cool gas passes now from the washer downward through pipe 16, and into the lower end of flue 8 surrounding chamber 4. It passes

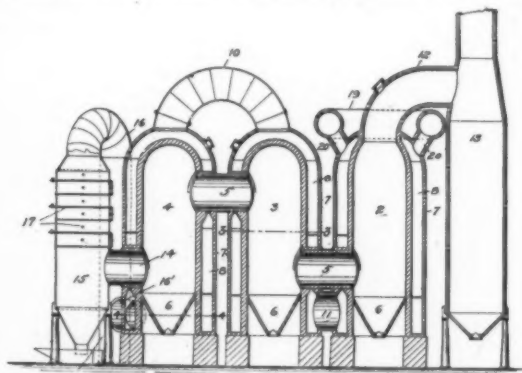


FIG. 4.—TREATMENT OF BLAST FURNACE GASES.

then upwards through 8 and through elbow 10 into the corresponding space surrounding chamber 3, and then through duct 11 into the space surrounding chamber 2. The pure gas is thus reheated, and its temperature when it enters the bustle pipe 19 is about the same as that of the impure gas coming from the dust catcher. The bustle pipe 19 has a series of connections 20.

Briquetting Friable Ores.—For the briquetting of natural friable ores and ore waste or dross, C. Reinke uses as binding agent so-called "Permian" limestone or other limestone con-

taining a high-percentage of calcium carbonate, an addition of a small quantity of magnesia and good Portland cement. About four parts of limestone are mixed with one of cement. The ore is intimately mixed with the mixture serving as binding agent, and the mass is then formed under great pressure into briquets. A special advantage is the low percentage of water in such a prepared briquet, analysis showing the following composition: Iron, 56.78 per cent; residue, 3.60 per cent; lime, 7.01 per cent; moisture, 0.55 per cent. The briquets thus produced stand red heat perfectly, and remain solid until removed from the furnace.

Self-Hardening Tool Steel.—J. Churchward (832,773, Oct. 9) patented a self-hardening tool steel of the following composition: 84 to 90 per cent steel, containing about 0.2 to 0.6 per cent of carbon; 4 to 6 per cent nickel; 0.5 to 1.5 per cent tungsten; 2.5 to 3 per cent chromium; 0.5 to 1 per cent manganese, and 0.25 to 0.5 per cent vanadium.

MERCURY.

Distillation Process.—In the ordinary method of heating ores by a current of fuel gas, the latter (*i. e.*, the total calories supplied to the furnace) is introduced at one end of the ore receptacle and passes through the ore in the direction opposite to the movement of the ore itself. The ore on its travel through the furnace is thereby gradually heated, while the temperature of the fuel gas current is gradually reduced. A radical departure from this practice is proposed in two patents of W. B. Dennis (833,679, process patent, and 833,680, apparatus patent, both of Oct. 16). In the Dennis process the furnace gases of the heat current flow through the ore-containing receptacle from the cooler portions to the hotter portions in the same direction as the ore is moved itself. Since the heat supplied from the gas to the ore in a given time depends on the difference of the temperatures of both substances, and since the daily capacity of the furnace (which is one of the chief factors in the cost of ore treatment) depends on the time required by supplying the heat to the ore, it would, of course, be impossible to supply the whole fuel gas at once at the cold end of the furnace, and let it pass together with the ore in the same direction through the furnace. For, in this case the thermal efficiency would be high only in the beginning, on account of the high temperature difference between the fuel gas and the ore, but the further the progress in the furnace the lower would be the temperature difference, and the less heat would be supplied from the gas to the ore in the unit of time. The principal feature of Dennis' furnace construction consists in dividing the whole furnace into a number of zones, and in adjusting and regulating the temperature slope between the ore and heat current in each zone according to the special conditions. The fuel gas made in the gas producer is not supplied in bulk at once at the cold end of the furnace, but is gradually supplied to each zone, together with enough air, according to the conditions prevailing in that zone.

The general scheme is indicated in Fig. 5. The different heating zones, eight in the present case, are 14, 15, 16, 17, 18, 19, 20, 21; the primary combustion chambers for the fuel gas are 32, 33, 34, 35 on one side and 40, 41, 42, 43 on the other side; the secondary combustion chambers are 44, 45, 46, 47 on one side and 36, 37, 38 on the other side; 39 is the final superheating chamber. The fuel gas, mixed with air, enters the primary combustion chamber 32, the amount of gas and air admitted thereto being regulated by valves. The mixture of gas and air passes across the furnace through the zone 14 to the secondary combustion chamber 44 on the other side of the furnace, and descends through the checkered floor thereof into the primary combustion chamber 40, into which an additional supply of fuel gas and air is admitted, which additional supply, mixed with the gases coming through the zone 14, travels through the zone 15, the result being that the temperature in the zone 15 is higher than it is in the zone 14, thus carrying out the primary principle of the invention, which is

that the fuel gas or heat current should pass from the cooler to the hotter portions of the furnace. This operation continues, the air and gas current flowing back and forth in opposite directions through the respective heating zones and being reinforced in each of the primary combustion chambers by an additional supply of fuel gas and air until it reaches the final heating zone 21, in which the temperature is the highest, and in cases of ore which is to be sublimed preferably above the boiling point of the metals contained in the ore to be recovered. Thence the current of air and gas passes into the superheating chamber 39, which incidentally is a dust-collecting chamber, and is connected with the dust collector proper. To insure the superheating of the final combustion chamber 39, provision is made for an additional supply of air and fuel gas thereto. This current of mixed air and gas is passed through the first cooling zone 22, the temperature in which is much less than in chamber 21.

The succession of the temperatures in the different zones is indicated by the following example: Supposing that the temperature of the gas current in the zone 14 is 200, and the temperature of the ore in said zone is 100, the temperature of

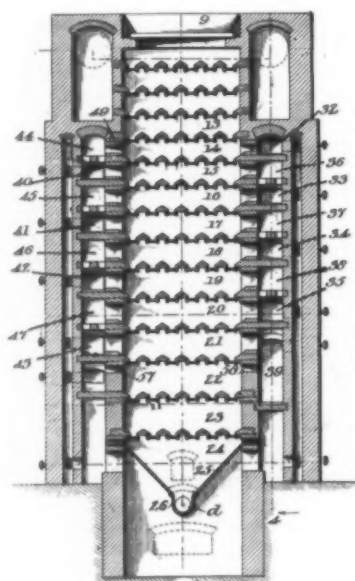


FIG. 5.—MERCURY FURNACE.

ing zones 22 the temperature is 1,700, all these numbers being in degrees on the Fahrenheit scale. These figures are given to illustrate the regulation of the temperature slope at a uniform ratio throughout the series, and are based on the assumption that it is desired to secure an ultimate induced temperature of, say, 1,700° in the ore before discharging from the last roasting zone, and that the temperature of the ore when delivered from the lowest drying chamber 13 into the first roasting chamber 14 is 100°, or that 1,600° of heat must be inducted into the ore.

The operation of the furnace is as follows: Assuming each zone of the ore tower to contain a charge of ore, and the ore in the last treating zone 21 to be fully treated, the grate-bars of the cooling zone 23 are opened, dumping the ore into the spent-ore pit 24. These grate bars are then immediately closed, and the bars of zone 22 opened and the ore dumped into the now vacant zone 23. This operation is repeated from zone to zone until the hopper 9 has been emptied and is ready for a fresh charge. Each series of bars is operated by a single lever, and these levers being located on the charging floor the entire dumping operation requires but a few minutes. The spent-ore being discharged by a spiral conveyor operated by power, the

furnace operator is not required to leave the charging floor during the entire dumping operation. In treating mercury sulphide ores the best results have been obtained by the inventor in practice by using a charge covering the ore floor from 6 to 9 inches deep, and a perfect roast has been secured in 4 hours. In a furnace with eight roasting floors or zones, as shown in the drawing, one charge would be dumped every 30 minutes. The dumping serves to thoroughly mix the ore better than could be done by mechanical appliances.

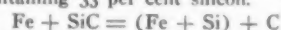
A distinctive feature of the process is the method of generating heat from fuel, the feature of which is that the fuel is oxidized gradually or by degrees in progressive zones of combustion by admitting to each zone a volume of air measured in proportion to the temperature of the fuel in such zone, and of causing the gases and vapors of each zone to pass into and through the next succeeding zone in the line of progression of the combustion process, the travel of the gases or products of combustion being always from a zone of lesser to one of more combustion, or from a cooler to a hotter zone. In the zone of incipient oxidation where the temperature is comparatively low, only water-vapor is liberated, and only sufficient air is supplied to produce this result. In the next zone of the series the temperature being somewhat higher than the preceding one, the lighter hydrocarbons are released, and sufficient additional air only is supplied to support this degree of combustion. In like manner from zone to zone the process of combustion is advanced gradually until complete oxidation has finally been accomplished in the last combustion zone. The gaseous products are then delivered into the reducing chamber, containing incandescent carbon through which the gases pass. No oxygen or air is admitted to this zone other than that already contained in the fuel gases. The well-known reaction which takes place here is the reduction of carbon dioxide and water to carbonic oxide and hydrogen.

The process is primarily intended for treating mercury ores, but is stated to be of much more general application, "as it may be used in connection with heating furnaces, or furnaces for desulphurizing copper matte, treating cement materials, or even in driers."

ANALYSIS OF CURRENT ELECTRO-CHEMICAL PATENTS.

Manufacture of Silicides, Ferrosilicon, Etc.—F. J. Tone, 833,427, Oct. 16. Application filed May 18, 1905.

The object is the production of various silicides, such as silicides of iron, copper, manganese, aluminium, calcium, or double silicides of these metals—as well as silicon alloys. The process may be described with respect to the manufacture of ferrosilicon. It has been known that considerable quantities of carborundum have been used in the iron and steel industries instead of ferrosilicon in order to introduce silicon into the steel, but a disadvantage of using carborundum is that at a temperature ordinarily obtained in the manufacture of steel, only limited quantities of silicon will be absorbed. It is therefore preferable not to use carborundum directly, but to use it as starting point for making ferrosilicon. For this purpose the inventor has devised a process in which the high temperature of the electric furnace is used. A typical reaction is the following, in which the charge consists of 56 parts iron and 40 parts carborundum, and there is formed a ferrosilicon containing 33 per cent silicon.



The carbon liberated in this reaction is preferably used as reducing agent for the simultaneous production of further amounts of silicon or iron. This is accomplished by adding to the charge corresponding amounts of the oxide of silicon or the oxide of iron. The following equation is typical of the process when iron oxide is added to unite with the carbon from the carborundum:



The following equation is typical of the process when silica is added for the purpose of uniting with the carbon liberated by the carborundum, thus producing a further quantity of silicon, which forms part of the alloy:



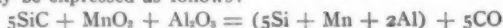
When silica is added to a mixture of the oxide and carborundum, the following equation is typical of the process:



If it is preferred to use with the carborundum only the oxide of the metal the silicide of which it is desired to produce, a mixture of the oxide and carborundum can be made such that the carbon liberated will be of the exact amount necessary to reduce the oxide, and a typical reaction may be expressed as follows:



The process may be applied also to the production of double silicides or to silicon alloys containing several metals—as, for example, an alloy containing silicon, manganese, and aluminium. For example, by using a mixture containing 200 parts carborundum, 87 parts manganese dioxide, and 102 parts alumina, there is obtained an alloy consisting of 140 parts silicon, 55 parts manganese, and 54 parts aluminium. This reaction may be expressed as follows:



Electrolysis of Fused Electrolytes in Induction Furnace.—

Leonard Waldo, 833,357, Oct. 16. Application filed June 23, 1905.

The apparatus is essentially a combination of the electric induction furnace with a cell for electrolyzing fused electrolytes. This is accomplished by arranging the electrodes at opposite points of the annular secondary of the induction furnace containing the fused charge. The current induced in the secondary from the primary is used to melt the charge, after which the primary circuit may be opened. The current between the electrodes need only be sufficient to produce the desired electrolytic action. In general, it will be necessary to close the primary circuit only intermittently, and then for no longer than is necessary to keep the charge at a temperature of fluidity. The main advantage claimed for the apparatus is that both the electrolyzing and heating currents may be controlled independently of one another, so that each may be adjusted to the particular needs of the moment.

Electrolytic Production of Hydrazo Derivatives.—O. Diefenbach, 833,513, Oct. 16. Application filed March 28, 1906.

The manufacture of hydrazo bodies, as hydrazobenzene and its homologues, by electrolytic reduction of the corresponding nitro, azoxy, or azo bodies suspended in the solution of an alkali or of an alkali salt according to the hitherto known process (see, for instance, the German Patents No. 120,899, Nov. 24, 1899, and No. 121,900, Dec. 1, 1900) shows the inconvenience that during the process the reduction products are precipitated in a solid form and are apt to stick to the cathodes and diaphragms so that they can only with difficulty be removed from the apparatus and at the expense of the durability of the latter. Besides, this coating or partial insulating of the cathode often causes a considerable rise of electric tension. To overcome these disadvantages, the present inventor adds to the electrolyte convenient solvents for the reduction products, which, however, are not themselves soluble in water, as for instance, benzene, toluene, etc. These solvents take up the reduction products, and at the end of the electrolysis they float on the electrolyte and can easily be removed from the cathode compartment, together with the reduction products, without removing the electrolyte itself. The following specific example is given: The cathode compartment of an electrolytic vat is charged with 100 cubic centimeters of caustic soda lye of 3 per cent strength and 100 grams of nitrobenzene, which are dissolved in 300 grams of benzene, while the anode compartment is charged with caustic soda lye of 5 per cent strength. The anodes and cathodes are of iron or nickel. The

cathode liquid is strongly agitated, and about 150 amp-hours are passed at a current density of 2 to 4 amps. per square decimeter of cathode surface, whereby about 80 to 85 grams of nitrobenzene are converted into hydrazobenzene, while about 15 to 20 grams only are reduced to azoxy benzene. Both reduction products dissolve in the benzene, and when the reduction is completed they can be decanted easily from the surface of the cathode liquid. By adding the necessary quantity of hydrochloric acid to the solution in benzene, the hydrazobenzene is converted into benzidine hydrochloride, which is readily soluble in water, while the azoxy benzene remains dissolved in the benzene, and therefore can easily be separated from the aqueous solution of benzidine hydrochloride. The solution of azoxy benzene in benzene is then returned to the cathode department, together with a fresh portion of nitro benzene. Any other hydrazo derivative and the corresponding benzidine derivatives may be converted similarly into hydrazo benzene and benzidine in the above manner.

Electrically Treating Air or Other Gases.—J. H. Bridge, 832,767 and 832,768, Oct. 9. Applications filed April 3, 1906, and May 25, 1906, respectively.

832,767 covers the process, and 832,768 the apparatus. The

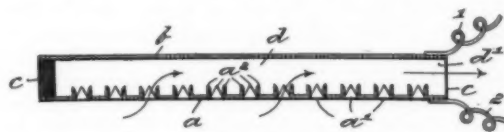


FIG. 1.—PRODUCTION OF OZONE.

principle of the apparatus is shown in Fig. 1, where *a* is a metal plate with openings *a'* surrounded by projections *a²*; *b* is another metal plate, while *c* is a U-shaped frame of insulating material, forming with the plates *a* and *b* a chamber *d* which has the outlet *d'*. By the connections 1 and 2, the two plates *a* and *b* are charged to opposite potentials. The air is passed through the openings *a'* and flows in the direction indicated by the arrows, being simultaneously subjected to the influence of electric discharges whereby the oxygen of the air is transformed into ozone.

On account of limitations of space, due to the convention report in our present issue, a large number of electrochemical patents had to be reserved for our next issue.

BOOK REVIEWS.

THE PRINCIPLES OF QUALITATIVE ANALYSIS, from the Standpoint of the Theory of Electrolysis Dissociation and the Law of Mass Action. By Wilhelm Böttger. Translated by William Gabb Smeaton. Philadelphia: P. Blakiston's Son & Co. Price, \$2.00.

Several years ago Ostwald's "Scientific Foundation of Analytical Chemistry" called the attention of the analytic-chemical world to the ionic theory, and showed that in some respects and within certain limitations, a clearer and better understanding of the analytical reactions is gained with this theory, as compared to the atomic theory. Böttger's book goes into all the necessary details and is a comprehensive work on qualitative chemistry.

While with the old theory the danger is prevalent that the analytic work is dissolved into a number of mechanical operations, without the necessity of thinking about the connection of the general facts with the special facts, this danger is removed by adopting the new theory (though it cannot be denied that another danger arises, namely, that of assuming more reality in some generalizations of the dissociation theory than there is in them in fact). On the basis of the dissociation theory, the student must learn the "why" of every reaction; he learns the reason underlying his manipulation according to the theory.

Bauxite Brick for Lining Rotary Cement Kilns.

In our February issue of this year (Vol. 4, page 52) we published an interesting paper by A. J. Aubrey on the refractory uses of bauxite in chemical and metallurgical work.

In this connection, some information recently made public on the use of bauxite blocks for lining the hot zones of rotary Portland cement kilns is interesting. The bauxite brick is made by the Laclede Fire Brick Manufacturing Co., of St. Louis, Mo. A test was carried out at the plant of the St. Louis Portland Cement Co. with the cooperation of Messrs. Struckmann and Taylor. On March 17, 1905, the hot zone of a 6-foot rotary, 60 feet long, burning pulverized coal, was covered with a 6-inch lining of bauxite. On Feb. 22, 1906, this lining was removed after a continuous run of eleven months and five days, or 8,208 hours, during which time at least 75,000 barrels of cement had passed over this lining.

On examination the blocks showed an average loss of about $2\frac{1}{2}$ inches, while a number of them had not worn much over 1 inch, and it is believed that the blocks would easily have lasted over the year if they had not been taken out.

The following features of bauxite brick make it suitable for cement kilns: First, bauxite blocks are basic, so that they are not chemically attacked by the cement mixture. They have a sufficient degree of hardness and porosity conducive to the adhesion of the cement coating, which affords protection from direct flames, and greatly insures its length of life. Further, bauxite has a very low heat conductivity, and therefore with about the same losses of radiation a 6-inch lining of bauxite brick can be used in place of a 9-inch fire brick lining. Finally, the life of the bauxite lining is twice that of the fire brick lining.

These advantages, of course, involve a very appreciable reduction in the cost of freight and cost of erection and maintenance.

Ball Filling Material for Sulphuric Acid Towers.

In an article in our Vol. II., page 347, we referred to the great progress which has been made in recent years in connection with the chamber process for making sulphuric acid, mainly as a result of the strong competition of the new contact process.

One of the most important details of a chamber plant is the filling material for the Glover and Gay Lussac towers as well as for the intermediate towers. Coke has been used to a large extent, but since its shortcomings were early recognized, earthenware material in brick and other forms were largely introduced. Attempts, however, continued to find a more suitable form of the earthenware filling, the main principle being to get as large a surface of reaction as possible.

To some extent the problem was solved by the introduction of the perforated hollow balls of Oscar Guttman in London, which in their outer and inner surfaces presented as large a surface of reaction as may be obtained in practice. But their disadvantage was that they had to be made by hand, and were, therefore, quite expensive. Nevertheless, it speaks well for their suitability as a filling material that they came into extensive use, especially in German chemical works, and chiefly in nitric and hydrochloric acid plants.

An important practical progress has recently been made by the Deutsche Steinzeugwarenfabrik of Friedrichsfeld, in Baden, Germany, who succeeded in producing the Guttman hollow balls by machinery. This means a very uniform manufacture and at the same time a greatly reduced price. They are now making balls of 100 mm. (4 inches) diameter with ten holes. This makes them available for the whole sulphuric industry.

In order to be able to give exact advice on the questions connected with the sulphuric acid industry, the Deutsche Steinzeugwarenfabrik has made an arrangement with Mr. Rudolf Heintz, in Hanover, to give such advice to all prospective users of the Guttman hollow balls. Mr. Heintz has given special attention to the subject for many years, and from a paper, which he published in the *Chemical Trade Journal*, Vol. XXXVIII., No. 992, the following abstract of his exposition of the advantages of the Guttman hollow balls is made:

The balls are moistened with a very thin layer of acid, both outside and inside. The gases enter the holes of the balls under a certain pressure and are at once diffused in them and brought into intimate contact with the moistened inner surface. Ball towers require no special work of erection. The balls are simply poured in, since they are equally efficient in any position, and the settling or deviation from the vertical of the tower, as well as the settlement of the balls themselves, does not influence the action in any way.

Heintz makes the following comparison of balls as filling material, with the best other kind of filling, namely, plates. In a preliminary Gay Lussac tower, which consists of twenty layers of plates, of sixteen plates each, the space occupied by plates and accessories has an area 27×27 meters. The plates being only 100 mm. apart, that the height is 2.3 meters and the cubical content is 16.77 cubic meters. Each plate has a surface of 0.6×0.6 meters, and the total contact surface of the plates is 230.4 square meters. The bearers have a surface of 76.8 square meters. Thus the whole tower has a contact surface of 307.2 square meters.

If we now fill such a tower with Guttman hollow balls of 100 mm. diameter, it will be found that to get the same contact surface as before 5,040 balls are required, and the volume required by them is 3.6 cubic meters, that is 4.7 times less than before. In other words, if it is desired to retain the same height of the tower of 2.3 meters (which may be advisable), the cross-section may be made 4.7 times smaller. Heintz shows that under such conditions there is no danger of an obstruction of the draft.

From a practical standpoint the reduced cost is of greatest importance. While coke filling is very much cheaper than the Guttman balls, yet the construction of the tower itself for coke filling is very much greater than for ball filling, so that it will be found on the whole that there is an appreciable saving due to the use of the Guttman balls, which at the same time are far more efficient.

Concerning the use of Guttman balls in different towers, Mr. Heintz makes the following remarks: Intermediate towers can, of course, be made in any shape or size, but the best rule is to make the height of that part which is filled with balls at least equal to the diameter of the tower. With the use of Guttman balls, Gay Lussac towers may be built with much smaller area and much inferior height. In fact, it will never be necessary to make them higher than 30 feet. The section of these two towers can be so calculated that the cubical content of the filled space amounts at the utmost to one-fifth of that in a coke tower. Thus, for example, a Gay Lussac tower of $2.4 \times 2.4 \times 12$ meters will be reduced to $1.48 \times 1.48 \times 7.50$ meters, and for this 15,475 balls will be required.

Except with sulphuric burners, Glover towers will never be filled in their lower part with balls, since these would fill up with mud by the dust. On the other hand, it will be undoubtedly advisable to use balls in about the upper third of the denitration zone. There is much time and space given on account of the good distribution of the liquid, and the shape of the balls allows a considerable temperature to be reached. For this reason also a smaller number of balls will be sufficient.

The Deutsche Steinzeugwarenfabrik, as manufacturer of the Guttman hollow balls, is represented in this country by Messrs. Fred. Bertuch & Co., of New York City.

Chemical Abstracts.

In accordance with the recent vote of the American Chemical Society the publication of a semi-monthly abstract journal will be commenced Jan. 1, 1907. The corps of those in charge of the various divisions of the journal is already well organized and work upon the abstracts has been commenced. It is intended to include in the journal abstracts of all new work in chemistry published in the world after Oct. 1, 1906. Chemical patents issued in the United States, Germany, France and England after July 1, 1906, will be included. The abstracts will be classified under the following divisions, the selection of abstractors and the oversight of each division being placed in the hands of the persons named:

Apparatus—W. H. Walker.
 General and Physical Chemistry—G. N. Lewis.
 Photography—L. H. Friedburg.
 Electrochemistry—W. R. Whitney.
 Radioactivity—H. N. McCoy.
 Inorganic Chemistry—Alexander Smith.
 Analytical Chemistry—L. M. Dennis.
 Mineralogical and Geological Chemistry—W. F. Hillebrand.
 Metallurgy—J. W. Richards, Henry Fay.
 Acids, Alkalies and Salts—T. L. Briggs.
 Glass and Pottery—G. E. Barton, Albert V. Bleining.
 Cements and Mortars—Harry Drew.
 Fuel, Gas, Coke—J. D. Pennock.
 Organic Chemistry—M. T. Bogert.
 Petroleum, Asphalt, Turpentine, Wood Products—S. S. Sadtler.
 Cellulose, Paper—A. D. Little.
 Explosives—C. E. Munroe.
 Dyes, Textile Fabrics, Bleaching, Inks—L. A. Olney.
 Pigments, Resins, Varnishes, India Rubber—A. H. Sabin.
 Fats, Fatty Oils and Soap—W. D. Richardson.
 Sugar, Starch and Gum—C. A. Browne, Jr.
 Leather, Glue—J. H. Yocum.
 Biological Chemistry—L. B. Mendel.
 Foods—W. D. Bigelow.
 Nutrition—C. F. Langworthy.
 Water, Sewage, Disinfectants, Insecticides—L. P. Kinnicut.
 Fermented and Distilled Liquors—Robert Wahl.
 Pharmaceutical Chemistry—A. B. Stevens.
 Soils and Fertilizers—F. P. Veitch, J. H. Pettit.
 Patents—W. H. Seaman.
 The dues of the Society for 1907 will be increased to \$8.00, to pay for the expense of the new publication.
 The abstract journal will be published twice a month.
 The *Journal* of the American Chemical Society will, of course, be continued and will contain original articles, book reviews and reviews of recent progress in the various fields of chemistry.

Notes.

Obituary.—We regret to report the death of Cav. MARIO MICHELA, President of the Società Italiana di Elettrochimica, in Rome, on Oct. 10. Through the courtesy of the late Cav. Michela we were enabled to give an illustrated description of their works in our Vol. I., page 453, September, 1903.

University of Pennsylvania.—The new building for the engineering departments of the University of Pennsylvania was dedicated on Oct. 19. The magnificent building is located at Locust and Thirty-third Streets, in West Philadelphia, and is splendidly equipped to meet all modern requirements.

Philadelphia Section of the American Electrochemical Society.—The first monthly meeting of the Philadelphia Section for the season 1906-7 will be held on Nov. 2, at Soula's Café, Fifth and Ludlow Streets, Philadelphia, at 8 P. M., preceded by an informal dinner at 7 o'clock.

Filter Papers.—We have received samples of ashless, chemically pure, toughened and starch-free filter papers made by Max Dreverhoff, in Dresden, Germany. The American representatives are George D. Feidt & Co., in Philadelphia.

Chemical Courses.—Among the chemical courses for "extension students," to be held at the Polytechnic Institute, of Brooklyn, College of Arts and Engineering, during the season of 1906-7, there are courses by Prof. Olsen on general chemistry, qualitative analysis, quantitative analysis and advanced quantitative analysis, and courses by Prof. Fay on organic chemistry and coal-tar dye.

Tropenas Converter Steel Process.—Messrs. Powell & Colne, of New York City, agents for the Tropenas converter steel process, have just closed a contract with the Massachusetts Steel Casting Co., Everett, Mass., for a second converter, after the first had proven very successful. Two other converters have been ordered by the Penn Steel Casting & Machine Co., Chester, Pa., and one by the Duquesne Steel Foundry Co., of Coraopolis, Pa.

Peroxides.—A pamphlet issued by the Roessler & Hasslacher Chemical Co. contains a reprint of a paper of R. von Foregger and Herbert Philipp, read before the New York Section of the Society of Chemical Industry, on earth alkali and allied peroxides; properties and applications. The authors deal with calcium peroxide, strontium peroxide, magnesium peroxide and zinc peroxide, and discuss the bleaching of oils, sterilization and preservation, the ageing of alcoholic beverages, sterilization of drinking water and of milk.

"Things Chemical."—The October issue of *Things Chemical* contains the portrait of Dr. J. E. Teeple, director of the Industrial Laboratories of the Charles E. Sholes Co.

Chemical Patents and Free Alcohol.—The October issue of the *Patent Review for Busy Manufacturers*, published by Messrs. Mason, Fenwick & Lawrence, of New York City, contains a list of chemical patents issued in the years 1889 and 1890, expiring in 1906 and 1907. They are classified under the following headings: Organic compounds, carbon lyes, dyes, dyeing processes, pharmaceutical preparations. The same issue contains a list of the important patents granted in the United States during the last twenty years for processes and apparatus for the manufacture of alcohol. This is specially interesting, since, as is pointed out, the free alcohol bill passed June 7, 1906, the provisions of which become effective Jan. 1, 1907, will have a tremendous influence upon chemical manufacturers in the United States, and especially upon the manufacture of many organic products now almost entirely imported from Germany. In the manufacture of dyes, transparent soaps, and in hundreds of processes where extraction with alcohol is necessary, the wonderful stimulus of free alcohol will be felt. Besides this, many new uses for alcohol in the household will be found and the production of suitable burners will also afford great scope to the inventor. Commissioner Yerkes is now in Europe gathering data to be used in the promulgation of rules for the manufacture and denaturation of alcohol.

Deoxidizing Alloys.—For some time past high percentage silico-spiegel alloys have been made in the electric furnace by the firm of George E. Blackwell Sons & Co., of Liverpool. The cheap grades are, first, about 70 per cent manganese and 20 to 25 per cent silicon; this grade, as a rule, contains about 4 or 5 per cent iron. Second, 50 to 52 per cent manganese and 20 to 22 per cent silicon, the balance being iron. Third, 36 to 40 per cent manganese, and 20 to 22 per cent silicon, the balance being iron. These alloys are largely used as deoxidizers in the manufacture of steel, but there has been a demand for perfect deoxidizers which contain no iron or carbon. The metallurgists of George E. Blackwell Sons & Co. have been working in this direction, and have succeeded in producing an alloy containing practically 30 per cent silicon and 70 per cent manganese. The content of carbon is less than $\frac{1}{2}$ per cent. The alloy is claimed by the manufacturers to be one of the most perfect deoxidizers yet introduced into the steel industry.

Coke Ovens.—The October issue of the *Proceedings of the Engineers Society of Western Pennsylvania* contains a paper by W. M. Judd on coke-oven construction. The paper is restricted to a description of the beehive type of oven, and is fully illustrated. Concerning the waste of heat in such ovens, it is said that a portion of the waste heat is now successfully utilized at some of the more recent plants. A very effective demonstration of the power wasted by the beehive oven is given at a plant recently completed, where, from the waste heat of fifty ovens all the electric power required, both for mining operations and at the coke ovens, for two plants aggregating 800 ovens is being generated at the present time; and at this same plant, within a short time, 100 ovens will furnish the power for the operation of five mines and coke plants aggregating 1,500 ovens. If a market for the power could be secured a further development of the use of waste heat for power purposes would doubtless bring about the making of other changes in the construction of the ovens.

Personal.

Mr. WOOLSEY MCA. JOHNSON, who has been acting for some time in a consulting capacity to Mr. Samuel W. Traylor, of the Traylor Engineering Co., is now also connected as metallurgical engineer with the Tri-Bullion Smelting & Refining Co., 43 Exchange Place, New York. This company owns three large mines, chief of which is the Kelly mine, Magdalena, New Mexico. The Tri-Bullion Smelting & Refining Co. intends to erect a 300-ton concentrating plant, producing 100 tons of high-grade zinc concentrates, as well as lead and copper concentrates. They will also erect either in the Kansas gas field or in the New Mexican coal fields its own reduction plant, to turn their high-grade zinc ores and concentrates into spelter.

THE BUSINESS of advertising the products of engineering concerns, heretofore conducted by Mr. H. M. Baxter, in Pittsburgh, Pa., will hereafter be carried on under the name of BROOKE & MCCONNELL, both gentlemen having been connected in the past with Mr. Baxter in the work of the firm.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

OZONE (Continued.)

No. 636,304, Nov. 7, 1899, Alexander Vosmaer, Haarlem, Netherlands.

The ozonizer comprises a metallic casing sub-divided into compartments, the walls of which constitute one of the electrodes. The other electrode or discharging surface comprises a large number of blades or points arranged parallel to these walls and in close proximity thereto. Both electrodes consist of steel, copper, brass, aluminium, or other suitable metals. The construction effects some economy of space as compared with one employing opposing blades or points, and there is said to be less liability of sparking. High-tension, continuous, or alternating currents are employed, and in some cases a condenser is arranged in shunt and an air gap in series with the discharge.

No. 636,868, Nov. 14, 1899, Henry Tindal, Amsterdam, Netherlands.

The ozonizer comprises a semi-cylindrical vessel or conduit of cast iron, to the inner surface of which is applied a thin layer of enamel, and a cover of plate glass suitably secured to the cast iron and supporting a plurality of electrodes, between which and the cast iron conduit the discharge occurs. These electrodes comprise semi-circular plates of steel arranged parallel with each other and concentric to the inner surface of the conduit. The air to be ozonized enters and escapes at opposite ends of the conduit, and all of the electrodes may be arranged at an inclination to the line of flow, in order to pro-

duce a whirling or like motion of the gas. It is stated that cooling is unnecessary, and that ozone of high concentration is obtained by passing air through the extended, even and dense field of discharge provided by the apparatus.

No. 640,694, Jan. 2, 1900, Marius Otto, Neuilly-Sur-Seine, France.

The discharge occurs between opposing points or edges, carried by the walls and partitions of a suitable cast iron casing. In order to prevent arcing between the points there is interposed between each pair of electrodes a rotating plate, the purpose of which is to interrupt the discharge at short intervals. Several constructions are described. In case the electrodes are in close proximity the rotating plate is of insulating material, having portions in the form of sectors cut away, the discharge being interrupted over any given portion of the electrodes as they are covered by the solid portions of the plate. In some cases the electrodes may be arranged beyond the normal discharging distance, and the rotating plate will then be of conducting material; in such case the discharge will occur as the solid portions of the plate pass a given portion of the field. In either case the sectors, instead of being entirely open can be provided with a very thin sheet of insulating material, thereby making a more complete baffle for the circulation of the air and increasing the efficiency of the apparatus.

No. 642,663, Feb. 6, 1900, Charles G. Armstrong and Wm. D. Neel, Chicago, Ill.

The ozonizer comprises a casing of insulating material, as glass, sub-divided into three compartments by two vertical insulating partitions. The electrodes consist of copper wires incased in heavy glass tubes, which are mounted in the central compartment in parallel relation and are supported by the partitions. The tubes carrying the electrodes of one polarity pass through one of the partitions, and are mounted in shallow recesses in the other, this construction being reversed for the tubes carrying the wires of opposite polarity. The wires constituting the electrodes of each polarity are brought into the respective lateral compartments and suitably connected. Perforated distributing plates are arranged at the inlet and discharge ends of the casing, the flow of air being transverse to the electrodes.

No. 648,764, May 1, 1900, Joshua H. Lamprey, London, England.

One electrode consists of a plate of metal covered on each side by sheets of dielectric material, as glass. The electrodes of opposite polarity are supported by these sheets, and comprise metallic wires having metal beads or discs strung upon them and spaced by insulating beads. The insulating beads are of larger diameter than those of metal, whereby contact of the latter with the dielectric is avoided. No casing is shown or described.

No. 653,078, July 3, 1900, William Elworthy, London, England.

A casing is sub-divided into two compartments of unequal size by means of a horizontal partition, the lower compartment being the larger. A large number of glass tubes, open at both ends, are so mounted in this partition as to project slightly into the upper compartment. A metallic rod is inserted in each tube, the several rods being connected together and constituting one electrode or discharge surface. The opposite electrode comprises a coil of metallic wire, cord or chain encircling the portion of each tube which projects into the lower compartment, being spaced from the tube by suitable means, as, for instance, a reversely coiled dielectric strip. Air is admitted to the upper compartment, passes through the several tubes, is deflected upwardly through the lower compartment and discharged at the upper portion thereof, thus passing twice through the field of discharge, once within the tube and once between the tube and the exterior coil. All electrical connections are made in the upper compartment, and are, therefore, not subjected to the action of the ozone.

